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THE

METALLURGY OF GOLD

A PRACTICAL TREATISE

ON

The Metallurgical Treatment of Gold-Bearing Ores

INCLUDING

THE PROCESSES OF CONCENTRATION AND CHLORINATION

AND

THE ASSAYING, MELTING, AND REFINING OF GOLD

BY

M. EISSLER

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SECOND EDITION, ENLARGED.

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PREFACE.

IN this volume my aim has been to present in a condensed form such an account of the several processes which are generally used for the Extraction of Gold from the Ore as shall be sufficient for the full information of investors, and others interested in gold mining operations, who may wish to gain an intelligent insight into the *modus operandi* at the gold mines. At the same time I have given full descriptions, with illustrations, of the machinery employed in those processes, in sufficient detail to make the book of substantial value to professional men and manufacturers concerned in the gold-mining industry.

The treatment of gold-bearing ores, although at one time so uncertain in its results as to be, not only a hazardous, but almost a purely speculative business, has now, through continued improvements in appliances and the adoption of more economical methods, developed into a settled industry, in which more or less profitable results may be obtained even from comparatively low grade ores.

When, forty years ago, the news was circulated throughout the world from California, that gold had been found in that territory by Mr. Marshall, in a millrace near Sutter Creek, thousands of enterprising adventurers flocked to the new Eldorado. So abundant were the glittering scales of metal which appeared in the gravel washes and in the rivers and tributary streams of the western slopes of the Sierra Nevada mountain ranges, that little or no metallurgical knowledge was required for the successful extraction of the precious gold dust. No machinery was then needed beyond a pair of stout arms, a pick, a shovel and pan—with perhaps a rocker and a few boards nailed together in a trough-like sluice; and it was with rude appliances such as these that many millions' worth of gold was taken from the Placer mines, thus laying the foundations for the Pacific empire.

The gold discoveries in Australia—which came a few years later—nearly outrivalled the rich Californian deposits, and gave a new lease of life to the Australian colonies; and to-day Great Britain looks with pride at those magnificent domains at the Antipodes, teeming with a large population and many flourishing cities, all of them owing their prosperity, in no small measure, to the discovery of gold and other metals.

From present indications the South African goldfields have also a brilliant future in store, but the discoveries in that region are too recent to afford means for the full appreciation of the possible results.

When gold was first discovered in California, the knowledge of metallurgy and mining in their several branches, among the people of the United States, was very rudimentary. But with that discovery a new era began over the vast expanse of territory extending from the Atlantic to the Pacific. The citizens of the great Republic awoke to the fact that in their mountain ranges were minerals of all kinds, representing enormous wealth, which could not be secured without extended knowledge of the requisite appliances. These were rapidly introduced in all branches of metallurgy, and in the advances thus made the treatment of gold-bearing ores has kept pace with the improvements effected in other branches of the science.

Brilliant results have been achieved in the last twenty years, and I shall rejoice if the present work assists in extending knowledge of the subject, and in further stimulating progress in invention and the development of gold mining.

LONDON,
August, 1888.

PREFACE TO THE SECOND EDITION.

IN preparing a new edition—which I am glad to find so soon required—I have been at some pains to supply additional matter on those subjects a fuller treatment of which has seemed desirable in the light of my experience with the first edition. The result is the enlargement of the work by about one hundred and fifty pages and forty additional illustrations.

My best thanks are due to the public press and private correspondents for valuable suggestions for a new edition; and of these I have availed myself wherever practicable.

There has been no interference in the present edition with the main plan of the original work, although five new chapters have been added. In these (amongst other matter) will be found much additional information as to machinery employed in crushing and amalgamation, and as to mills in actual operation. Later developments of the process of Chlorination are also described, and a chapter has been added on the Smelting of gold ores and the refining and parting of gold bullion. I have also given some account of the physical features of the gold-producing districts of North-Western America, which I hope will be none the less interesting because of the element of personal experience which is there introduced. In the first chapter of the work I have inserted an account of the interesting operations of Hydraulic Mining, with which the history of many gold-producing districts is so largely concerned.

In its enlarged form I trust the work may prove increasingly acceptable and useful to those for whom it is designed.

17, BELSIZE CRESCENT,
SOUTH HAMPSTEAD, LONDON, N.W.
April, 1889.

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THE
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CHAPTER I.

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I.—Gold Mining in California.

After the exhaustion of the easily worked "placer diggings" of California, the miner turned his attention to quartz. He soon found that the appliances then in existence in other parts of the world for the treatment of gold-bearing rock did not correspond to the local conditions of the country. Improvement after improvement of those appliances was introduced, and in a few years the now perfect Californian gold mill was the result of the ingenuity of the western millmen. Prior to 1861 quartz-mining operations were merely in an experimental stage, but about that time the quartz lodes, which were the original sources from which the enormous placer deposits derived their wealth, began to be seriously mined and systematically attacked, with results that have kept California in the foremost place of the gold-producing regions of the world, and she will, no doubt, through her numerous quartz lodes, continue to occupy the position of a gold producer for many generations to come.

As a rule, quartz mining in California was at first carried on upon the basis of "free milling"—namely, taking the ore out of the mine, crushing and pulverising the same in the battery, and extracting the gold by amalgamation on copper plates. Only a very small percentage of the gold in the ore was saved in this manner, the quantity of gold which can be secured by the agency of quicksilver being only a fraction of the assay value of the whole mass, while by far the largest portion of the gold went to waste, carried away by the water which flowed from the battery and got lost in the tailings. Many failures in gold mining were the result of the inability of the millman to "catch the gold."

But it was not long before these lessons of experience brought about the introduction of new methods, and quartz mining is now assuming the character of a scientific industry; so that ores which twenty-five years ago would have been thrown into the waste heap as "base" and "rebellious," are now carefully piled up, taken to the mill, and treated with great profit.

In the year 1866, like thousands of other mining adventurers, I was engaged in "placer" mining operations in Idaho Territory, a country whose name at that time was little known even in the United States. In company with some friends I discovered a lode, which we worked at first very successfully; but as we got to a greater depth our difficulties began, owing to the increase of iron pyrites in the ores, and the decrease of the "free gold." I then undertook a journey to Grass Valley, California (which in those days meant a journey in a stage coach of nearly one thousand miles, the great trans-continental railway not being then completed), where the first quartz mines were being opened up and worked on "scientific" principles. Messrs. Watts were then the owners of the celebrated Eureka Mine, and some Cornish miners, Coleman by name, were prospecting the extension called the Idaho. The Colemans are now millionaires, and the Idaho one of the best mines in California. I was kindly permitted to investigate the works, and it was with no little interest that I studied the first processes of Californian gold extraction. These processes, in the course of eighteen years' residence in many mining districts, I have seen in development from their infancy. A Mr. Deetkin had erected the first reverberatory roasting furnaces and chlorination works for working sulphurets, and a portion of the sixth chapter of this work is devoted to an account of his operations as described by the late Professor Guido Küstel. At the Eureka mill I found the concave buddle in operation for concentrating the rich slimes; the Eureka rubber, erected by Mr. Mellville Attwood, a renowned English mining engineer; the Attwood amalgamator (also named after him), and other improvements. These appliances were subsequently described by Mr. R. W. Raymond in his mining reports to the United States Government, and his descriptions of the apparatus will be found reproduced in this book. They were the first in use, and are still in operation.

The invention of other concentrating machines and grinding pans have, since that time, enabled quartz miners to improve the yield of ore; and among others the Frue vanner, Duncan's concentrator, and Hendy's concentrator, deserve to be mentioned. The construction of the mortars has also been perfected.

We have been taught that the first condition involved in the successful extraction of gold from the matrix is to gather and save the sulphurets and other metallic combinations in which it is held, and from which the various amalgamating processes have thus far been powerless to extract the gold. As only a certain percentage of the gold in quartz mines is present the free state, the present millman, to make gold quartz mining a success, must depart from the old beaten path of "free milling" and amalgamation. We must look to new processes and new methods to make mines pay. The lodes consisting of pure quartz and free gold are getting scarcer and scarcer every day; and although they are occasionally found in new districts, yet after a certain depth is reached this free milling character changes. This involves a systematic procedure by which "low-grade" ores can be cheaply and quickly transmuted into high-grade ores, and when such transmutation has been accomplished they are ready for scientific metallurgical treatment, by which, through the aid of chemistry, they may be converted into ingots of pure metal, the base metals with which it is combined being at the same time brought into use and made to yield their proportion of profit to the intelligent miner.

The new processes and new methods by which these important results are to be brought about mean, first, simple and cheap methods of concentration—appliances by which, when low-grade ore has been mined, it may be quickly and cheaply denuded of the large part of its useless or waste materials (its *gangue*), the *free gold* it contains caught by quicksilver and amalgamated, and its other valuable mineral ore particles concentrated and treated by chlorination or otherwise.

It is a well-known fact that in most gold-producing countries where the operations for the extraction of gold from quartz veins have been carried on, the "brown" ores, as they are called, or the true "free milling" ores, are generally comprised in that portion of a gold-bearing lode or vein which lies above the perpetual water-line proper. They are that portion of a mineral vein from which the rebellious elements have been oxidized, leached out, or eliminated by influences which have been acting through a long course of years, leaving the gold free and clean, and readily susceptible of amalgamation by and with quicksilver in any of the various methods which have been adopted for its use for that purpose. Below the perpetual water-line these gold-bearing veins generally assume the form of gold carried in sulphurets or other combination, to which the application of the amalgamating process has always proved unremunerative.

The loss in working gold ores, even with our most modern appliances, is still enormous, and even at the present day not more than one-half of the gold contained in the ore which is worked, whether by free milling or otherwise, is saved.

One of the oldest and best authorities in California (Mr. A. B. Paul) has said upon this subject: "As far as California is concerned, I am satisfied that no more than 40 per cent. of her gold is exhausted. The fact is, we are not working to save gold but to crush rock." Again he says: "Our present general system of gold mining is based upon the idea that gold is mainly coarse, while examination will show that the high percentage is in atoms finer than flour itself. In my experiments gold has been taken up so fine that in distilled water it would not precipitate in less than from five to ten minutes. Can you save gold of this kind by running water down stream? Again, can you obtain gold of this kind without minute reduction? Therein lies the secret of high assays before working and small returns after." Or, as a mining authority once remarked, "The gold is so fine in the rock that it takes a bushel of it to make an ounce."

Mr. Deetkin, of Grass Valley, in order to determine the loss of gold by the mill process, has tabulated a series of assays made of the tailings of ore of the best mills in the State, which shows the loss to have been 40 per cent. of the yield, of which the float loss was nearly 14 per cent.

In Australia, during eighteen months' milling of 85,521 tons, and daily assay of the tailings, the yield being only $6 52c., or £1 7s. per ton, the average loss was found to be $1 56c., or 6s. 6d. per ton, or 24 per cent. rated upon the amount saved. Add to this the amount of float gold carried away in the water, and the total loss would not fall short of 40 per cent.

The actual average yield of ores milled and smelted, calculated from Fossett's table of seven years' work in Colorado, showed the average value of the ore by assay to be $37 97c., or £7 18s. per ton. The average value per ton saved by milling and smelting was $14 50c., or about £3; showing a loss of $23 47c., or about £4 17s. per ton in gold and silver, or more than 60 per cent. The gold caught on the copper plates is, under the most favourable circumstances, only 50 per cent. of the assay value of the ore. The gold from the blankets and in the buddle concentrator does not amount to more than 5 per cent., so that when treating the most tractable of these sulphurets, battery amalgamation does not secure more than 55 per cent. of gold.

At the Merrifield mine in Deer Creek, Nevada County, a large proportion of the ore is heavily impregnated with sulphurets, and notwithstanding the fact that the rock by fire assay showed its value to be hundreds of dollars per ton, yet after working the quartz by the old process and in the most approved manner, including the use of Frue's concentrator, it was found that the average pay of the ore was about £5 per ton, including sulphurets. They are now satisfied that nearly two-thirds of the gold in the rock was lost by being carried off in fine particles with the sediment and water.

Professor Egleston says: "In all the methods for the extraction of the precious metals there are considerable losses. With the perfection of processes the main object is to reduce them, or else to cheapen the labour of extracting the ores. These losses are greater than is usually supposed, because as a general rule systematic assays of the tailings are not made."

It was also found that water from the mills three-fourths of a mile below them contained in suspension, on an average of twelve assays, nearly one pennyweight of gold per gallon. There were in this locality 516,000 gallons of this water flowing away in twenty-four hours, or a loss of about £70 per day. It was estimated that the annual loss of two mills working 250 days in the year was £17,000. From these and similar data the conclusion is drawn that the loss is between 40 and 50 per cent. of the total yield of the ore.

The State of California, since the discovery of gold on Sutter Creek in 1848, has produced in gold over £250,000,000; and it is estimated that more gold has been wasted in milling and hydraulic mining, and lost to mankind, by being washed down the rivers and partly carried to the ocean, than has been thus produced.

A happier state of affairs exists in the silver regions. In Nevada I have worked ores which yielded in silver from 88 to over 90 per cent. of their assay value, and it is to be hoped that in another generation inventions will have been perfected by which a higher percentage of gold may be saved.

From what has been said above it will be seen that the problem of gold milling is not the easy matter which some may imagine, but that there is a wide field open for investigation. Much has no doubt been accomplished, but there is plenty of room for improvement yet in the methods and appliances for securing the much-coveted metal.

II.—Gold as a Metal.

From the earliest times gold has been esteemed the most valuable of all the metals. It is not only distinguished for its rareness, but it possesses many very valuable properties not common to the other metals. It is especially valued for its richness of colour, and the fact that its brightness is not liable to tarnish.

The specific gravity of gold varies between 19·258 and 20·72, and its fusing point has been determined at from 2,200° to 2,596° Fahr.

Gold in the pure state is very soft, like lead, and may be easily scratched by the nail. It is more malleable and ductile than any other metal, and is capable of being beaten out into leaves $\frac{1}{280000}$ inch in thickness, while one grain of it can be drawn out into a wire 167 yards long. Its tenacity, however, is inferior to that of iron, copper, platinum, and silver. Its specific gravity is inferior only to that of platinum and a few of the rarer metals, such as iridium and osmium. In its natural state gold is almost always found alloyed with other metals;

the few compounds found being *sylvanite*, or telluride of gold and silver; and *nagyagite*, or telluride of lead with gold, silver, and copper.

Gold is known to exist in very small quantities in galena, in which, according to Percy, it occurs probably in the form of sulphide.

The distribution of gold is very general throughout the world, but at the present time the largest quantities are obtained in Australia, the United States, the Ural Mountains, and in parts of Africa.

It is almost always found in a metallic state; generally in small grains or scales known as *gold dust*; sometimes in particles so minute as to be invisible; and occasionally in pieces of considerable weight, called *nuggets*.

It is doubtful whether when gold occurs in pyrites it exists in every case in minute metallic particles, or whether in some cases it may not be present in combination with sulphur. The auriferous pyrites, as Dumas remarks, contain gold scattered throughout their mass in such small quantities that it is almost impossible, even with the aid of the microscope, to ascertain in what state the precious metal exists. As roasting the auriferous ore is considered useful with a view to its subsequent amalgamation, a doubt may be entertained whether gold is really present in the metallic state, notwithstanding that this has been generally supposed to be the case. Dumas, however, thinks that the powerful electro-negative tendency of sulphide of gold affords a strong presumption in favour of the hypothesis that this metal may exist in iron and copper pyrites partly, or even entirely, in the form of double sulphide. Brongniart observes, that it is chiefly when in combination with these sulphides—as also with galena or sulphide of lead, blende or sulphide of zinc, and mispickel or arsenical pyrites (a sulphide of arsenic and iron)—that the gold becomes invisible to the eye, a circumstance which is regarded as strongly confirming the opinion expressed by Dumas.*

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* **My own opinion is that gold in pyrites is not in a state of chemical combination, but is present in the metallic state, coated by sulphur, arsenic, &c.**

The other minerals with which gold is found associated are grey cobalt, lithoidal manganese, native tellurium, malachite, sulphide of silver, red silver, and sulphide of antimony.

Gold is occasionally, though rarely, found in a crystallised state, in which form it is of great value.* The most common forms are those of the cube and octahedron, or some modification of these forms. The faces are generally dull, and in most instances the edges are slightly rounded.

It is sometimes crystallised by cooling, but when such is the case it is, I believe, more by accident than design, as it is impossible to regulate the rate of cooling. I have myself frequently cast ingots of gold which, when chipping off the corners for assay purposes, I found to be in a crystallised state. I made it a rule to remelt such ingots, and found that on adding a little nitre a tough metal was obtained. From this I concluded that the crystallisation was the result of some impurity in the gold; though I never sufficiently investigated the subject to be sure on the point.

Composition of Gold.—Native gold is never quite pure, being almost invariably alloyed with silver, and containing frequently small proportions of copper and iron. In Siberia it is often associated with platinum, and in the Gongo Saco Mines, in Brazil, an alloy of gold and palladium of a pale yellow colour is sometimes found. In Columbia a mixture somewhat similar is obtained, the palladium being replaced by another rare metal called rhodium.

The proportion of silver, which is the principal metal found in alloy with gold, varies from 1 to 50 per cent.; and it not only differs greatly in specimens of gold obtained from different localities, but differs even in specimens from the same district.

Properties of Pure Gold.—Gold does not combine directly with oxygen, and therefore suffers no change on exposure to air and moisture at whatever temperature; not even if it be kept in a state of fusion in open vessels. Neither is it attacked by mineral or simple acids, except selenic, and then only by the aid of heat.

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* It becomes valuable, that is, for use as specimens—not intrinsically—gold crystals being highly prized by collectors of minerals.

The alkalies have ordinarily no action on gold. If, however, the gold and the alkalies together become heated during exposure to the atmosphere, an absorption of oxygen takes place, resulting in the formation of an alkaline aurate. Nitre when in a state of fusion attacks it. The persulphides, whether in the wet or the dry state, also attack it, resulting in the formation of sulphide of gold. Iodine has a weak action upon it, while bromine and chlorine at ordinary temperatures easily attack it. It is dissolved by any substance which liberates chlorine; it is dissolved therefore by hydrochloric acid, if binoxide of manganese or chromic acid be added to it.

The conductivity of pure gold for heat and electricity is little more than half that of silver. If the conductive force of silver be taken as 100, we have for gold—

Conductivity for heat	53·2
Conductivity for electricity at 27° F. . .	55·19

When in a state of fusion gold exhibits a bluish green colour. It is not sensibly volatile in the strongest heat of a blast furnace; but under the focus of a large convex lens, in the intense heat of the oxyhydrogen jet, or under the influence of a powerful electric discharge, a gold wire is dispersed in vapour; and if, in this latter case, the wire be placed just above the surface of a sheet of paper, the course of the discharge is marked by a broad, dark, purple stain, produced by the finely divided gold. If instead of the sheet of white paper a plate of polished silver be employed, it is traversed by a brightly-gilded line, which is firmly attached to its surface. When a globule of gold is placed between the terminal charcoal points of a powerful voltaic battery, it becomes almost immediately fused, and gives off abundant metallic fumes.

The contraction of gold on cooling is very great; it cannot therefore be advantageously employed for castings.

Gold-bearing Ores.—These consist chiefly of quartz, and in some cases they contain slate, baryta, and talc. I remember one occasion when gold was found in limestone. The precious metal is usually scattered through the rock in small particles, sometimes so minute as to be scarcely visible to the naked eye. Sulphides of iron, lead, copper, and zinc, as well as arseniurets, are frequently found with the gold. In pyrites the gold appears to exist in a metallic state, as it can be separated in some measure by grinding in the amalgamating pan.

The amalgamation of gold has been brought to a certain state of perfection by Californian millmen; and their methods of treating the gold-bearing ores have been very generally adopted in Australia and other countries. These methods are known as the "Californian stamping process," and this work is mainly concerned with an account of the several operations comprised in it, and of the machinery and appliances required in working it; but a description may be appropriately given here of the "hydraulicking" operations which form so interesting a feature in the history of gold mining.

III.—GOLD WASHING BY THE HYDRAULIC PROCESS.

By this process, or "hydraulicking," is understood the removal of large masses of gravelly soil from its original *locus* to another place of deposit by the impact of water under a very high pressure.

In California and Australia there are large deposits of this gravelly soil containing gold, and extending throughout whole mountain ridges, but not in sufficient quantity to pay, unless the washing be carried out on a gigantic scale. As a rule, these alluvial gravels, such as constitute the hydraulic mines, are not rich throughout the upper strata, and the gold is usually found concentrated in layers near the strata reposing on the country rock, designated in mining parlance "bed-rock." In some instances the bottom stratum of gravel is so rich in gold and the upper layer (which in cases reach a height of 200 to 300 feet) so poor, that it would prove unprofitable to remove the whole mountain mass, and this led to the working of the rich gravel beds by means of shafts and tunnels, whereby only the bed-rock stratum is excavated and washed, and the remainder left standing. Such mining is called "drift-mining."

In hydraulic mining, the whole depth of the deposit down to the bed-rock is washed away by means of powerful jets of water. The water disintegrates the soil, which in some cases is very hard, and during its onward journey through bed-rock tunnels and sluices submits the soil to a thorough system of comminution and trituration, which liberates the free scales and small lumps of gold. These sink by their superior specific gravity in the sluice boxes, and are collected therefrom after a run of thirty or forty days of such continuous washing.

Such a process requires a large volume of water, which has to be brought in canals or ditches, or aqueducts, from the fountain heads located in the mountains at a suitable elevation. The construction of these water conduits involves very large outlays of money and labour, as some of them extend from forty to one hundred miles in length.

The next point to be considered in hydraulicking is the removal of the large quantity of detritus resulting from the washing away of the huge masses of gravel in which the gold has to be sought—from the removal, one might say, of whole mountain ranges. But when once the proper application of the great mechanical force furnished by a large volume of water under high pressure was grasped, it was seen that the problem could only be satisfactorily solved if the auriferous gravel could be carried to some distant point and there discharged into a rapidly flowing stream. For accomplishing this object a hydraulic mine has to be provided with a tunnel or "open cut" from the nearest and most suitable ravine or river bank, so that, starting in the bed-rock, on the face of the ravine, or other selected point, it shall approach the centre of the gravel mass to be moved at a gradient of one in twenty-four to one in thirty-six. The dimensions of the tunnel are usually six feet in width and seven feet in height, sometimes wider.

The tunnels vary in length from a few hundred feet to a mile. The end of the tunnel is designed to reach beneath the under surface of the gravel in the centre or deepest part of the channel, at a point where a shaft or incline is sunk from the surface through the gravel until it intersects the tunnel. It obviously demands careful engineering to carry out works of such magnitude with the accuracy required, and, for the want of sufficient care or skill in this particular, years of costly labour and anxious expectation have been wasted in the early history of these enterprises.

It is therefore necessary in laying out an hydraulic mine operation to take precautions, by an accurate survey, (1) to ascertain the head or fall of water and its quantity; (2) to secure an outlet for the washed débris or tailings; and (3) to strike the lowest point of the channel with the bed-rock flume or tunnel. Outlet, dump, and fall have to be secured for the many millions of cubic yards of gravel displaced, and many valuable mines have had to suspend operations from their inability to get rid of the accumulated débris, which had "banked up" at the outlet of the flume (or water course) and stopped further washings.

When the tunnel has been completed and connection made with an opening in the gravel deposit, the tunnel becomes a sluice way, through the whole length of which sluice boxes are laid, to direct the stream and save the gold. For this purpose a trough of strong planks is placed in the tunnel, from three to four feet wide, with sides high enough to control the stream. The pavement is usually composed of blocks of wood from six to eight inches in thickness, cut across the grain of the wood, and so placed as to expose the end of the blocks to the wear of the current. The wooden blocks are usually alternated with sections of stone pavement, the stones set endways. In the interstices quicksilver is distributed, as much as two tons of this metal being required to charge a long sluice. The location and inclination of the sluices have an important

INTRODUCTORY CHAPTER.

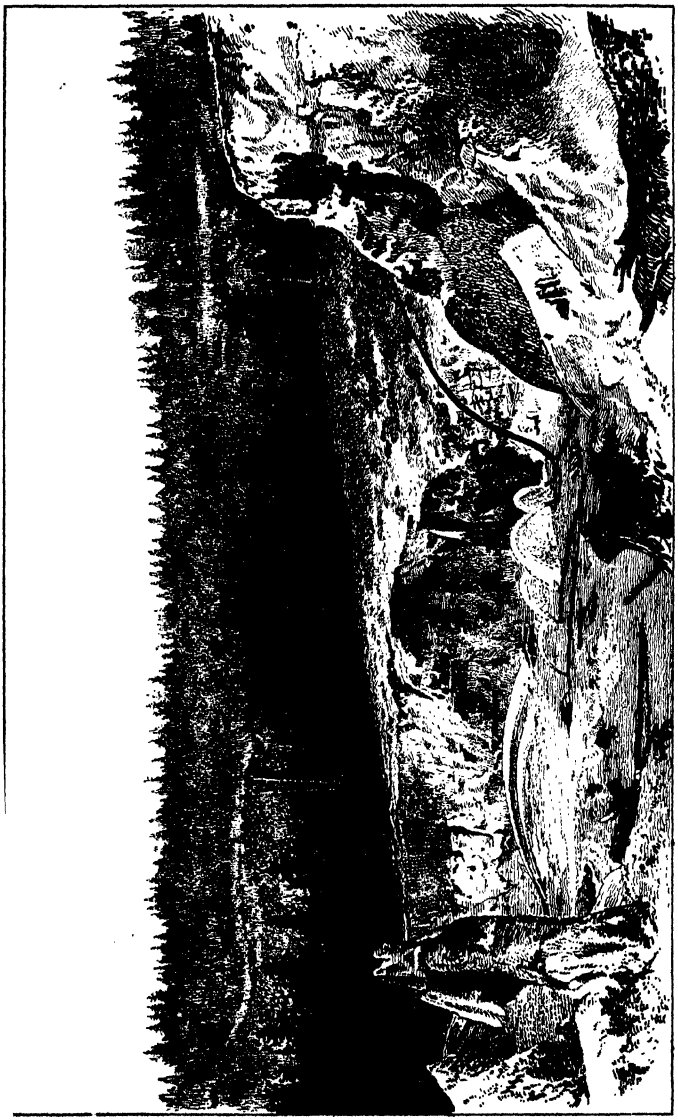

FIG. 1.—HYDRAULIC MINING ON THE GEORGETOWN DIVIDE, EL DORADO COUNTY, CALIFORNIA.

bearing on the results which are obtained in gravel washing and the proportion of gold saved, and proper attention has to be paid to the grade of the sluice boxes.

I will now revert to the removal of the gravelly soil by the force of the jets of water brought to bear upon it. The water is conveyed to the mining claim from the ditch or canal by means of iron pipes, the point of the supply being from 200 to 500 feet above the point of discharge. These iron pipes have to be carefully laid, and are often sustained on an incline of timbers. The water can of course be carried in pipes over depressions in the ground, and over intervening elevations, and so across valleys and mountains. The iron pipes are of great strength; they are made riveted at the joints, and measure from 12 to 20 inches in diameter. A pipe of ⅜ inch wrought iron can sustain a pressure of 385 pounds per square inch.

The pipes are connected with a powerful apparatus of sheet iron provided with a universal joint, to which the outlet or "nozzle" is attached, ending in a steel ring for the delivery of the water, which varies from 4 to 8 inches in diameter, as greater or less volume of water is used. From these nozzles the streams are directed against the face of the gravel "bank" to be washed with a force comparable to that of ordnance. The volume of water employed varies, of course, with the work to be done, but it is not uncommon to see four such streams, each conveying 300 to 600 inches of water, or more, acting simultaneously on the face of the same "bank." The accompanying illustration (Fig. 1) depicts this stage of the operations. One thousand "miner's inches" of water are equal to 106,600 cubic feet of water per hour, constantly discharged under a pressure of 100 to 200 pounds to the square inch, varying with the height of the column, or pressure. Under the continuous action of this enormous mechanical force, aided by the softening influence of the water, large sections of the gravelly mass come crashing down with great violence.

The débris, speedily dissolving and disappearing under the resistless force of the torrent of water, is hurried forward in the sluices, precipitated with the whole volume of now turbid water down shafts, or dumps, prepared for the purpose, and then taken up again by the sluices, where it is brought by means of undercurrents, riffles, and other appliances in contact with the quicksilver, and thus made to give up its gold. Boulders of 100 to 200 pounds in weight are shot forward by the impetuous stream, together with masses of the harder cement, which meet, in the fall and concussion of the great boulders, the crushing agencies required to disintegrate them.

The deeper banks of gravel are usually worked in two benches. The upper is never so rich as the lower, and, being also less firm, is worked away with greater rapidity. The lower section is much more compact: the stratum on the "bed-rock," being strongly cemented by sulphuret of iron and great pressure, resists even the full force of the water stream, until it is loosened by gunpowder. For this purpose adits are driven in on the bed-rock 40 to 70 feet from the face of the bank, and a tunnel extended at right angles therefrom to some distance each side of the adit. In this tunnel a large quantity of gunpowder is placed, from one hundred to one thousand kegs, and fired as one blast by fuse or electricity. In this manner the compact conglomerate is broken up, and the water then rapidly completes the work. This disintegration of the gravel banks by means of explosives is called by miners "bank blasting."

The sluices are often made double, for the convenience of "cleaning up" one of them while the other is in action. The process of cleaning up is performed, according to the extent of the works and the richness of the material washed, at intervals from fifteen to thirty days, and consists in removing the pavement and blocks from the bed of the sluice, and gathering the amalgam of gold and "rich dirt" collected in them, and replacing the blocks in the same way as at first. Advantage is also taken of the opportunity to reverse the position of the blocks and stones forming the pavement, where they are worn irregularly, and to substitute new ones for those worn through.

The mechanical action of the washing process on the blocks is, of course, rapid and severe, so as to require a complete renewal of them once in eight or twelve weeks. Some miners prefer a pavement of egg-shaped stones set like cobble-stone pavement, the gold being deposited in the interstices. Most of the sluices, however, are paved with rectangular wooden blocks, with or without stones as described above.

Rude as this method of saving the gold appears, experience shows that more gold is won by it than by any other method of washing yet devised; while the economical advantages it offers are incomparably greater. In fact, it would be impossible to handle so vast a body of material in any other way.

Among the noticeable features of California are the muddy rivers, which largely owe their turbid condition to the operations above described. These minor streams discharge into the Sacramento, which in its turn empties into the Bay of San Francisco, building up great bars with its burden of gravel. The damage done to farming lands in the lower portions of the Sacramento valley was, indeed, so great from the accumulation of débris, that legislative measures had to be taken to compel the hydraulic miners to impound their "tailings" and not discharge them into the rivers, with the result that many valuable mines had to stop working.

CHAPTER II.

TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION.

Section I.—MECHANICAL PREPARATION OF GOLD ORES.—Rock-breaking Machines—Blake's Patent—Reduction of the Ore by Stamps—Arrangement of a Battery described—Feeding the Batteries—Tulloch's Ore Feeder—Operation of the Stamps—The Capacity of a Stamp Mill.

Section II.—AMALGAMATION OF THE FREE GOLD.—The Attwood Amalgamator—The Hydrogen Amalgam Process—Grinding Pans—The Knox Pan—The Wheeler Pan—The Soderling Pan—Settlers and Rubbers—The Eureka Rubber—Treatment of Amalgamated Copper Plates—Silver-plated Copper Plates.

Section III.—THE PROCESS OF AMALGAMATION.—Operation of the Mill—Capacity and Regulation of a Stamp Mill—Professor Raymond on the Operation of Crushing—Aprons, Sluices, and Blankets—Water required for Gold Milling—Amalgamation in the Battery—Conditions of a Good Result.

I.—MECHANICAL PREPARATION OF GOLD ORES.

BEFORE subjecting the ore, or the auriferous gangue, to the chemical or metallurgical process by which the gold is extracted, it has certain mechanical operations to undergo, the number and nature of which will be determined by the character of the ore itself, and its state of aggregation. Nature—unfortunately for the operations of the metallurgist—has distributed this most precious metal in the rocks in a very fine state of division, the rock being impregnated with it in the shape of dust, spangles, or small grains, which on the whole are difficult, very difficult indeed, to separate; and it is only in extremely rare cases that gold is found in nuggets which require only to be cast into ingots. Occasionally rich specimens are found in which the separation of the gold is easily effected.

A large proportion of our gold has been, and is still, obtained by the direct washing of alluvial deposits. This book, however, only treats of the extraction of gold from rocks or ores, and I begin the present chapter with an account of those operations by which the rock is artificially reduced to the state of an auriferous sand or pulverised ore.

The treatment of gold-bearing rocks comprises the following operations:—

1. Crushing it by means of rock-breakers and stamps, and amalgamating the free gold inside and outside the battery by various contrivances.

2. Concentrating the free gold lost during crushing, and amalgamating it in pans with the auriferous sulphurets; and

3. Extracting the gold contained in pyrites by roasting and chlorination.

The ore—which in California is always called "quartz," no matter what the gangue may be—is first passed through a rock breaker, where the large lumps are reduced to pieces about the size of a walnut or even smaller, and is discharged upon a platform immediately in front of the battery of the stamping mill, or into self-feeders, from which the quartz is fed very regularly into a large oblong, rectangular, cast-iron mortar, in which five immense pestles play up and down, pulverising the quartz into very fine particles. As these pestles weigh from seven hundred to a thousand pounds apiece, in an establishment (called a mill) which works from twenty to sixty stamps, the noise is deafening, and the mill is not exactly the place for nervous persons.

After the quartz is pounded fine, we wish to get rid of the worthless waste rock, and for this purpose a stream of water flows into the mortar which dilutes the fine material, and while it is held in suspension by the water, it is splashed by the agitation caused by the stamps through an oblong slit on one of the longitudinal sides of the mortar, which is closed by a sieve. The fine particles of rock form with the water a very thin mud, which is called slime or pulp.

The question now arises, What becomes of the gold? The particles of gold which have been liberated during the process of trituration, either—by reason of their greater specific gravity—sink to the bottom of the mortar, where they come into contact with mercury and become amalgamated; or they are splashed against the sides of the mortar—which are lined inside with amalgamated copper plates—and are there "caught;" or they are splashed out of the mortar through the sieve along with the slime.

On emerging from the mortar the slimes pass over an apron of amalgamated copper plates, by which means a considerable quantity of metal is removed, as gold will adhere to these plates, and is thus prevented from passing down the sluices which immediately succeed the apron. The sluices are covered with blankets, and are set on an incline, and the gold and heavy ore particles collect between the fine hair of the blankets. These are washed every few hours in large wooden tubs, and the stuff collected on them—called "concentrations"—is treated in amalgamators, where the free gold is extracted.

The slimes, after passing the blankets, and also those from the amalgamators, discharge into the Eureka rubber, in which any particles of gold which have escaped are cleaned and brightened by rubbing, and are to some extent collected by means of amalgamated copper plates. The impoverished sands and slimes, after passing from the blankets, sluices, rubbers, &c., are discharged into concentrators, in order to collect small particles of gold enclosed in pyrites and other metallic sulphides, which would otherwise be lost. How the concentrations are treated will be subsequently described. In some mills the amalgamating apparatus is differently arranged.

Having given the rationale of the general method of dealing with gold-bearing ores, I will now enter upon a description of the various machinery and processes employed.

Before it can be submitted to the final pulverisation in the stamp mills, it is necessary that the quartz should be broken into small fragments somewhat uniform in size, and for this operation powerful rock-breaking machines are employed. Many satisfactory contrivances have latterly been introduced for economical and effective rock-breaking, but Blake's machine (which I will describe in detail) is most extensively used.

Rock-Breakers.—*Blake's Patent.*—This machine is shown in the illustrations (Figs. 2 and 3) in vertical section and perspective view. In Fig. 3, A *a* is a heavy frame, cast in one piece, with feet to stand upon the floor or on timbers, which frame supports all the other parts; *b* is the fly-wheel, one on each side, the shaft of which is formed into a crank; *c* is a pulley on the same shaft; *d* is a rod connecting the crank, *e*, with the lever, *f*: this lever has its fulcrum on the frame, *g*. A vertical piece, *h* (a front view of which is shown at B), stands

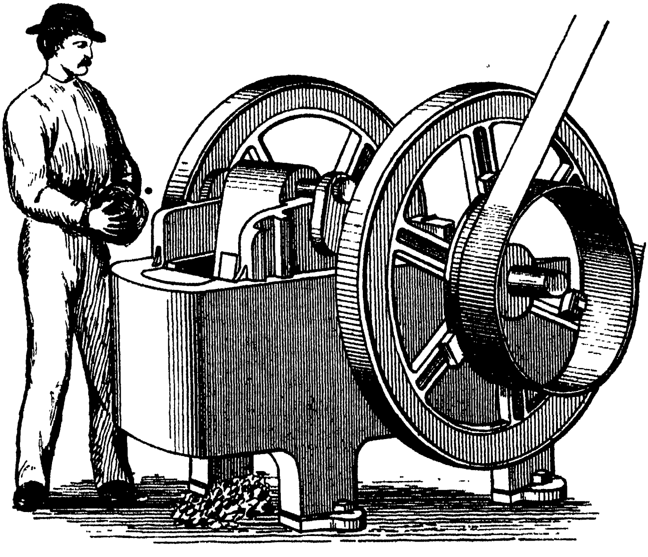

FIG. 2.—BLAKE'S ROCK BREAKER. Perspective View.

upon the lever, and against its top the toggles, *i i*, have their bearings, thus forming an elbow or toggle-joint. K is a fixed jaw against which the stones are crushed. The jaw is held in its place by keys, *l*, which fit in recesses in the interior of the frame on each side; *m* is a movable jaw, faced with a corrugated die, *m'*. This jaw is supported by a round bar of iron, *n*, which passes freely through it and forms the pivot upon which it vibrates; *o* is a spring of india-rubber, which, being compressed by the forward movement of the jaw, aids its return. C is the lever viewed from above. E is a top view of the die, *k*, which is corrugated on one or on both sides. In the latter case it is turned after one side is worn out; *p* is a screw by which the wedge, *r*, can be screwed up, in order to bring the jaw, *m*, nearer to *k*, and thus effect a finer crushing.

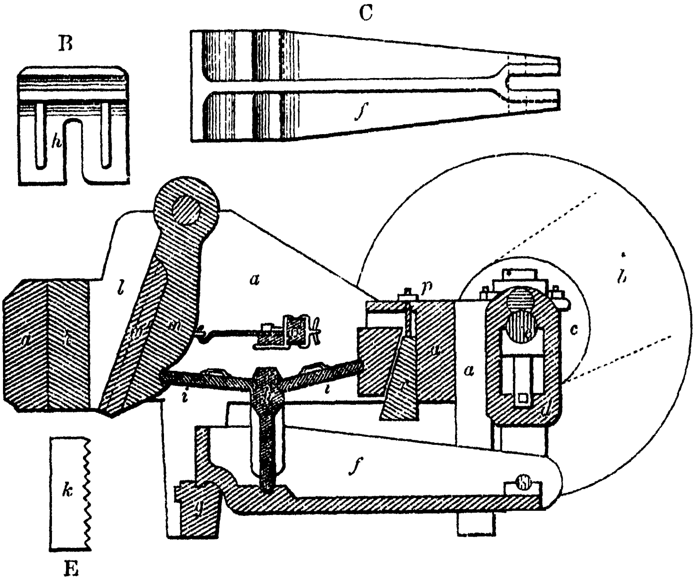

FIG. 3.—BLAKE'S ROCK BREAKER. Vertical Section.—Scale ½ in. to 1 ft.

The crank, *e*, revolves from one hundred and fifty to two hundred times per minute. The distance between the jaws may be raised to five-eighths of an inch by the screw, *p*. Other variations may be made by changing either or both of the toggles for longer or shorter ones.

Capacity of Rock-breakers.—A rock-breaker, of suitable dimensions, making 170 strokes of ¾ of an inch a minute, is capable in twenty-four hours of giving a preliminary crushing to 72 tons of quartz, which is the crushing capacity of a heavy 30-stamp mill. By preparing the rock for the battery, the rock-breaker increases the crushing capacity of a mill by 20 per cent.

Only the coarser quartz is passed through the breaker; the finer, which always contains a quantity of wood splinters brought from the mine, being crushed in a separate battery by itself. The splinters lessen the efficiency of the battery by clogging the screen holes, which latter require, therefore, a greater amount of attention to keep them in order.

Reduction by Stamps.—No machine effects the reduction of ores so well as the stamping mill. The construction of stamp mills varies very much, especially in their foundation frames, but in their main points they all agree, the work done by stamps being in principle the same as that of a pestle and mortar. There are generally five crushing stamps employed in one mortar. In California one mortar with its stamps is called a "battery." The frame of the battery is generally made of timber.

Arrangement of a Battery: the Frame.—For the purpose of erecting a battery a pit is sunk to the solid rock, the bottom of which is nicely levelled off for the reception of the mortar blocks, *a a*, Fig. 4. The block has a section of 3 by 5 ft., and rarely exceeds 10 ft. in length. It is set on end, and consists of two logs, 36 by 30 in. each, firmly bolted together with 1¼ in. bolts, and connected also with dowels 18 in. long, and 8 by 8 in. in section, introduced 2 ft. from each end. The horizontal section of the pit has usually a space of 24 in. all round the mortar block, filled up to the level of the mud sills with hydraulic concrete to within 4 to 5 ft. of the top of the mortar block.

After the top of the block has been carefully riveted, levelled, and planed, it is ready for the reception of the iron mortar, which is fastened to the block by means of 1¾ in. key bolts, *k k*, passing through the bottom flanges of the mortar (Fig. 4).

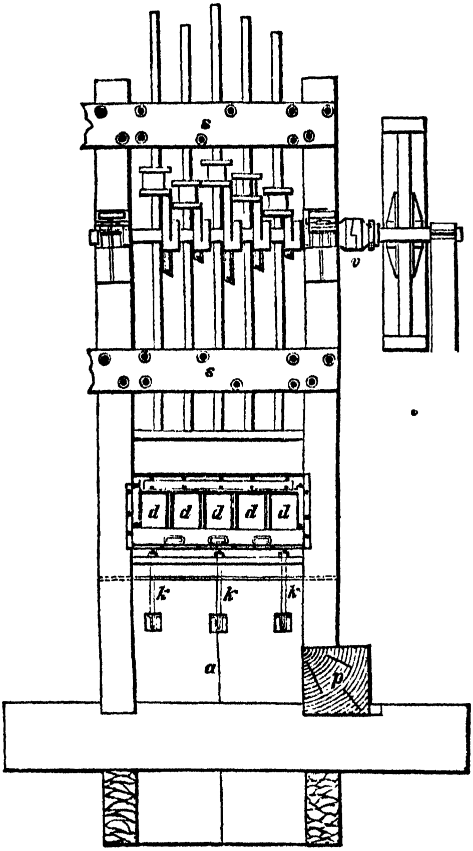

Fig. 4.—Front View of Battery.—Scale ⅛ in. to 1 ft.

The wooden battery frame which carries the cam shaft is independent of the mortar block. The woodwork is usually of sugar pine. The mud sills, *l l* (Fig. 5), 24 by 24 in., and usually three or four in number, are laid parallel with the cam shaft. The sills on each side of the mortar block are secured by iron bolts, which pass through the hydraulic concrete and have an iron anchor plate underneath. (See Fig. 5.)

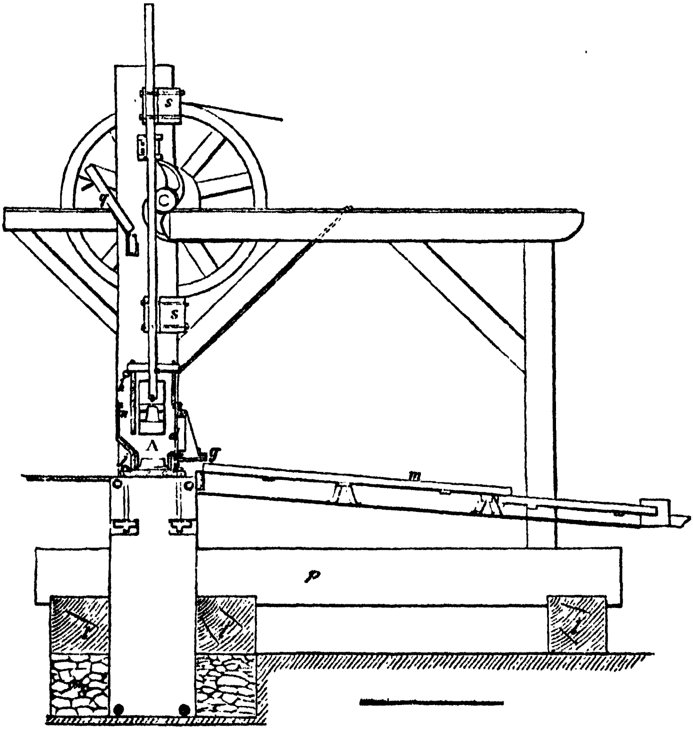

FIG. 5.—TRANSVERSE SECTION OF BATTERY. Scale ⅛ in. = 1 ft.

Cross sills, *p p*, 18 in. wide by 24 in. deep, and 18 ft. long, which are at right angles to the mud sills, carry the main posts, 21 by 24 in. in section, and to these posts the cam shaft is attached. The cross sills are let and keyed into the posts in the manner shown in Fig. 4.*

The Foundation.—When the ground is marshy or otherwise unreliable, deep excavation is not desirable, a broad base only being required for the frame foundation, as shown in Figs. 6 and 7. The surface of the ground must be dug out from 1 to 3 ft. in depth, and with as much width and length as the condition of the ground and the number of batteries may require. The illustrations show a foundation having a horizontal mortar block for two batteries. Fig. 6 is a front view and Fig. 7 a side view. Flooring underneath the timber with boards is improper, and in case the bottom appear unsafe the lower timbers must be laid close together. The bottom timbers, *a a*, are 12 in. square and 18 ft. long, and are laid for the required distance upon the carefully levelled ground. Between *a a*,

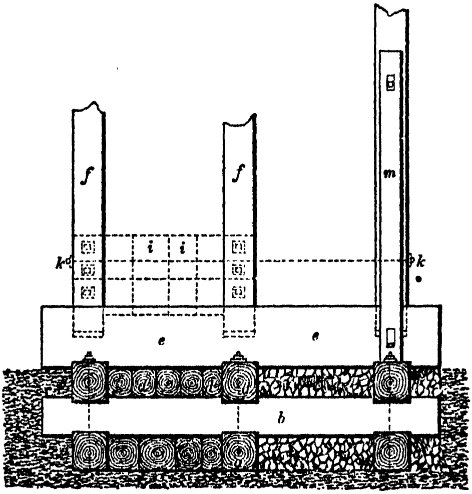

Fig. 6.—Elevation of Battery Frame. Scale $\frac{1}{8}$ in. = 1 ft.

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* An illustration of an iron battery frame (20 stamps) will be found at p. 100.

in the centre, short pieces, *c*, about 2 ft. long, are laid, so as to fill up the space under the double cross timbers, *b′ b″*. There are generally as many beams, *a*, *a*, *a*, as there are uprights, *f*.

The cross timbers, *b b*, are laid upon *a*, and over *b* a row of beams, *d*, corresponding with *a a*. The mortar block, *e*, completes the horizontal frame. The latter is 20 to 30 in. square, and its end is secured by an iron band. Oak is of course the best material for these blocks, but owing to its comparative scarcity, pine blocks are more generally used. Their length naturally depends on the number and size of the batteries.

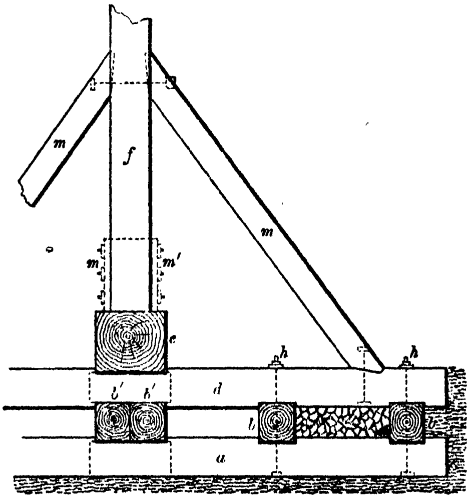

FIG. 7.—SECTION OF BATTERY FRAME. Scale $\frac{3}{10}$ in. = 1 ft.

To give greater firmness to the foundation framework the timbers are fastened together by bolts, *h*. The space between them is filled and beaten in with clay, loam, or, still better, with coarse gravel and stones, as shown in the figure at *g*.
the ground becomes consolidated at a depth of four

or five feet, the whole frame is set so much deeper, in which case short vertical pieces (as indicated by the dotted lines, *i*) are fitted to the mortar block and bolted together lengthwise (as shown at *k*), whilst their lower ends are let into the block. The planks, *m'*, prevent lateral displacement of the blocks, *i*.

Horizontal mortar blocks are also advantageously used on hard rocky ground, in which case, however, excavation is both difficult and expensive. In this case too the use of less timber is necessary. The ground in this case may be considered an artificial stone block, as represented in Fig. 8. The mortar block lies on the beams, *b b*, and these latter rest on the bottom pieces, *a*.

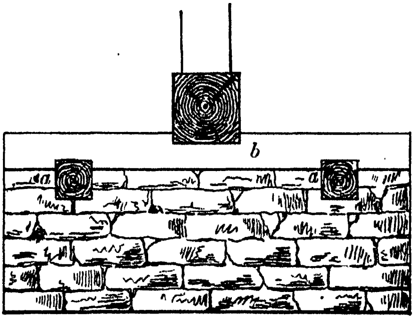

FIG. 8.—STONE FOUNDATION FOR BATTERY. Scale $\frac{3}{16}$ in. = 1 ft.

It contributes much to the steadiness of the frame if the bottom timbers are anchored to the rocks with iron rods; but in case heavy stamps are used, a foundation such as is shown in Fig. 8 will, if the space between the timbers is filled with rocks, be found perfectly satisfactory.

Instead of wood braces iron rods are frequently used, or iron and wood together, as shown at *m n* in Fig. 9.

The advantage of iron rods is that they take up less room than wood braces, and are capable at the same time of being, if necessary, tightened. The advantage of wood braces is that they insure greater steadiness than iron rods. Fig. 10 shows the wooden frame of the battery in elevation.

Mortars.—Fig. 12 represents a mortar having a single discharge, and designed for copper plate lining back and front, the former bolted through the mortar and the latter fastened to a block under the screen frame. The die, shoe head, and a part of the stem are shown at rest in the mortar. Fig. 11

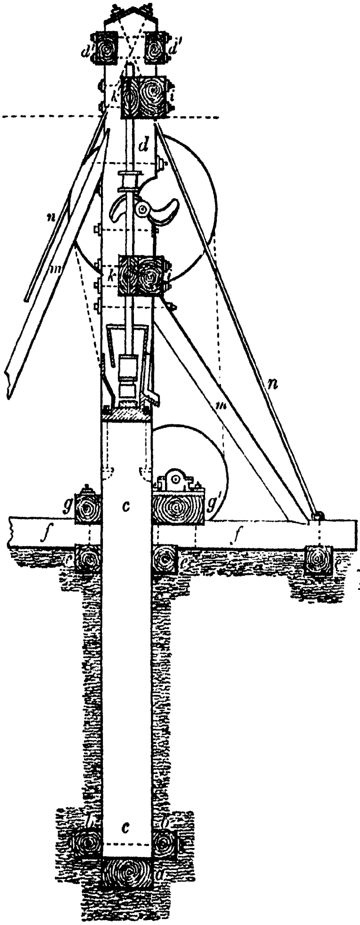

FIG. 9.—ARRANGEMENT OF STAMP STEMS.

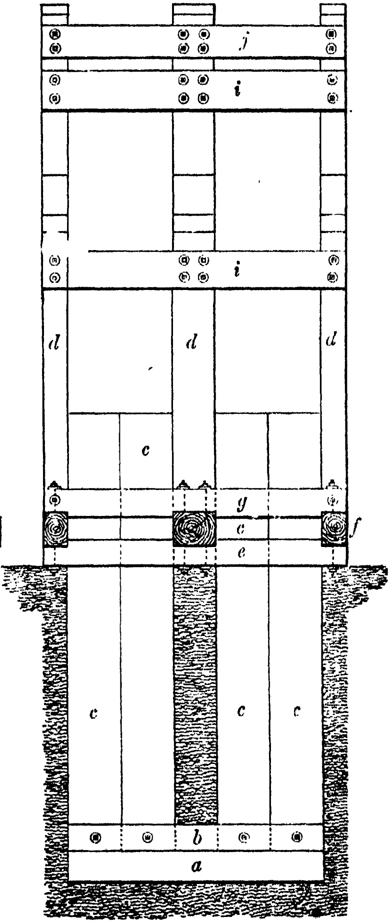

FIG. 10.—COMPLETE BATTERY FRAME.

Scale $\frac{1}{8}$ in. = 1 ft.

represents a mortar having a single discharge, and designed for copper plate lining in front only. It is especially adapted for crushing base ores needing concentration and fine granulation in order to separate the mineral from the rock. The discharge of the pulp is regulated by the fineness of the screens.

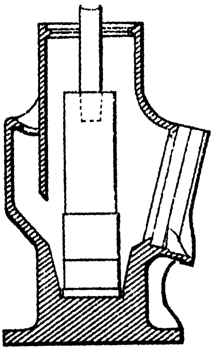

Fig. 11.—Mortar for Front Lining.

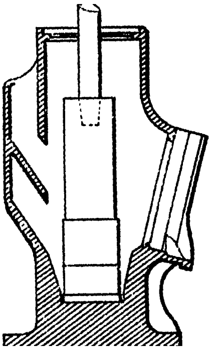

Fig. 12.—Mortar for Front and Back Lining.

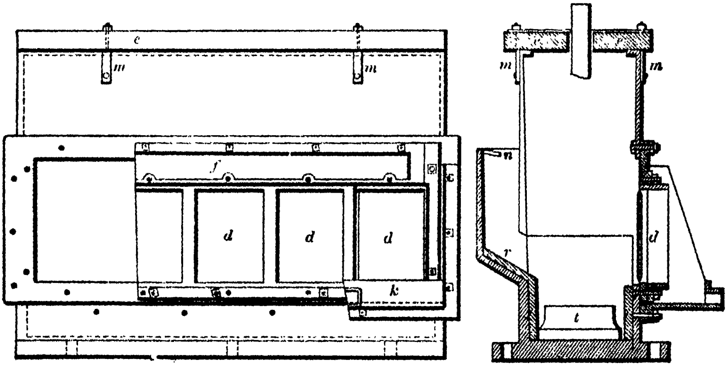

Fig. 13.—Mortar with Sieve Frame. Scale ⅜ in. = 1 ft.

The mortar (Fig. 13) consists of a single casting 3 ft. 9 in. high. The bottom is usually 3 in. thick, but sometimes more.

The walls are vertical on the outside, 1½ to 1¾ in. thick on the bottom, tapering to ½ in. on the top. The bottom flanges for fastening the mortar to the block are 2½ in. thick and 4 in. wide. In order to prevent the premature destruction of the mortar near the dies, cast-iron plates, *r*, 1 in. thick and about 24 in. high, are placed as a lining all round the inside, which can be replaced when worn out. These being wedge-shaped at the lines of contact with each other do not need any additional fastening.

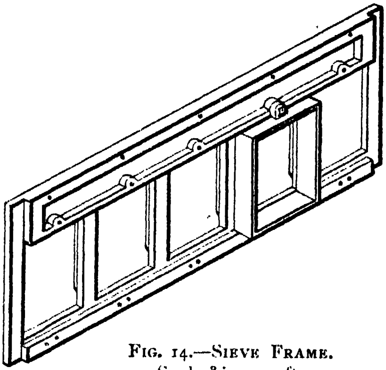
Fig. 14.—Sieve Frame.
Scale ⅜ in. = 1 ft.

At one of the sides of the mortar, opposite the discharge opening, *d*, is situated the feed slit, *n*. It is 3 in. in width and from 6 to 12 in. below the top of the mortar, and extends across the whole width of the mortar. A rim 3 in. wide is planed off round the discharge opening, *d*, for the purpose of receiving the sieve frame (Fig. 14) and the splash box (Fig. 25), both of which are of cast iron, and are bolted to it. The horizontal mortar bed receives the dies, *t*, the bottom or foot plates of which fit almost close to the sides of the plate lining of the mortar.

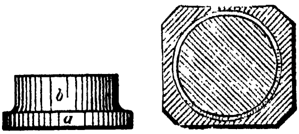

Fig. 15.—The Die.
Scale ⅜ in. = 1 ft.

The Die.—The foot plate, *a*, of the die (Fig. 15) is of a rectangular form, and its corners are cut off for the purpose of facilitating its removal in cleaning the battery. It is 1½ in. thick, and carries a cylindrical piece, *b*, 3½ in. high, the diameter of which is usually 10 in. When worn down to the foot plate it is replaced by a new die. Each battery consists of five stamps, which when new are all of a uniform weight.

The Stamp.—The stamp consists of four cylindrical pieces: the head or boss, the stem, the tappet, and the shoe.

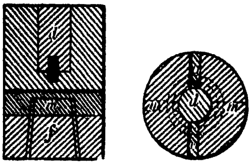

Fig. 16.—The Head.
Scale ⅝ in. = 1 ft.

The Head (Fig. 16) is of the same diameter as the shoe, and is from 15 to 20 in. long. Its lower face is provided with a conical recess, *f*, 6 in. deep, which receives the shank of the shoe, and a tapering core, *d*, to receive the foot of the stem.

To facilitate, in case of breakage, the detachment of the shoe and stem, channels, *m m*, *o o*, are provided below it for the insertion of wedges, which pass easily through the boss.

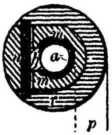

Fig. 17.—Stem and Tappet.
Scale ⅝ in. = 1 ft.

The Stem (*a*, Fig. 17) is of wrought iron, and varies in length from 13 to 15 ft., and in thickness from 3 to 3¼ in. It is turned true in the lathe. Both ends of the stem are tapered off, so that either may serve as a foot.

The Tappet (*r r*, Fig. 17) is in the main a cylinder of cast iron, from 9 to 11 in. in diameter, and 10 in. long, having a central bore. It is slipped on the stem and fastened to it by means of a wrought-iron gib, *b*, keyed up by wedges, *n n*, at right angles to the gib. The projection, *o o*, prevents the slipping of the gib, which would cause an uneven wear of the cam face. The gib is introduced into the mould previous to casting.

The tappet is slipped on the stem, and wedged above and below by means of the wrought-iron gib, *b*, which is 2 in. wide, and may be as long as the tappet, or shorter. Its inner face forms a part of the circular hole which receives the stem. N *n* are the key seats, into which the gib, *b*, slightly projects, so that if a key be driven into the seat it will press the gib against the stem, and so fasten the tappet to it. The tappet may in this way be easily moved to any point which the wear of the shoes may from time to time make necessary. The tappet when worn on one side may be reversed, as both sides are alike. The use of too much grease or oil prevents the revolution of the stamp, and, as a consequence, not only does the surface of the tappet become damaged, but the advantage of a uniform wear of the shoe is destroyed.

As every part of the working face of the tappet becomes worn by the cams—with the exception of a concentric ring ½ in. wide round the stem (the cam passing within half an inch of the stem)—an annular recess, *m m*, 1 in. deep, and concentric with the stem bore, but 1 in. longer, is bored out on the working faces to prevent the wear of the edges of the cam. Both faces of the tappet are used as working faces, and when they are worn down to the extent of an inch they are replaced by new ones.

The Shoe (Fig. 18) is generally made of white iron, cast wholly in sand and slowly cooled; but it is sometimes made of steel. In the Californian stamp mills the form of the shoe is generally round.

The cylindrical part or butt, *a*, is 5½ in. long; the shank, *b*, being one-half the diameter of the butt, has the same length, and tapers upwards conically. The diameter of the butt varies from 9 to 11 in. The head is fastened to the stem by laying two strips of thin cloth crosswise over the stem head, and then driving the stem home. The tappet is then keyed fast to the stem, and the shank of the shoe, which is surrounded by small wooden wedges pointed upwards and held in position by a string, is set vertically under the shank hole of the boss. In falling, the stamp wedges the shank of the shoe firmly into the shank hole, after which two or three blows are given at the top of the stem with a sledge hammer. The battery is then allowed to play slowly, so that the stamp may be lifted six or eight times while the others are at rest. The shoe is not allowed to strike on the naked die; a piece of board or plank is therefore placed between the shoe and the die. The lining formed by the strips of wood must have sub-

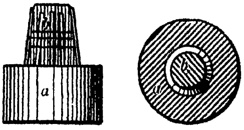

FIG. 18.—THE SHOE.
Scale ⅝ in. = 1 ft.

stance enough to prevent the edge of the head coming in contact with that of the shoe. The shoe should be renewed when worn to the extent of an inch.

Weight and Power of the Stamp.—The weight of stamp varies from 600 to 900 lbs., and is generally between 750 and 850 lbs. The weight is 10½ lbs. per square inch of crushing surface. The shoes are placed 1 in. apart, and 1 in. from the linings of the mortar. The proportional weight of the stem, head, shoe, and tappet, respectively, to the total weight of the stamp is as 40 : 29 : 16 : 15 : 100. The weight of the die when new is usually $\frac{16}{100}$ of the weight of the stamp.

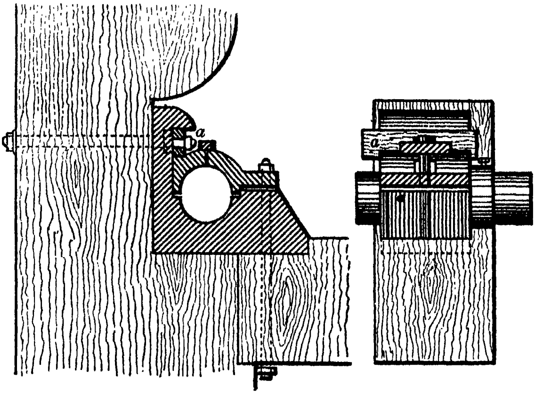

FIG. 19.—THE JOURNAL BOXES.—Scale ⅝ in. = 1 ft.

The effective mechanical power per second of the stamps in a battery is found by dividing the product of the number of stamps, the weight of one stamp, the lift in feet, and the number of lifts per minute, by 60 seconds. The power applied on the shaft must be greater than the effective power, on account of the friction of the stamps in the guides, as well as of the cams and tappets and the shaft journals, and also on account of the jarring of the whole machinery, which practice has determined at from 1½ to 1¾ horse-power per stamp, according to the weight of the stamp.

The Cam Shaft is of wrought iron, and generally 5 in. in diameter when turned in the lathe. It is provided with two key ways, 1 by ¾ in., which for the purpose of insuring the normal position of the cams are placed at right angles to each other. The centre of the cam shaft is usually 5 in. from the centre line of the stems. The shaft is placed 9½ ft. from the mortar bed.

The weight of the stamps should be distributed on the shaft as equally as possible. It will be found an advantage to power and machinery to have one shaft for several batteries, as in this case if there be, say, thirty or forty stamps in the mill, they can be all arranged on the shaft in such a manner as to insure that no more than one stamp shall catch the tappet at the same time, in which case the shaft will bear the same weight at each moment of the revolution. If the same number of stamps were driven by, say, six or ten separate shafts, each connected with a main shaft, this latter shaft would have to bear from four to six times as much weight at one moment as at another, depending of course on the time of starting the separate batteries.

On the other hand, there are reasons which make it inadvisable to use one shaft for several batteries. For instance, the stamps may each be easily and independently arrested in the several batteries; but should a cam get loose, all the batteries will have to be idle while the repairs are going on; while still more inconvenience would arise if one of the cams should break, as in this case all the cams in the way must be first removed. For this latter reason cams fitted directly to the shaft are not so good as those screwed or wedged on rings or collars. All things considered, a middle course would seem to be the best. Two five-stamp batteries with one cam shaft are better for the machinery than two shafts.

The position on a shaft of the cams for two five-stamp batteries is shown in Fig. 20. A is a section of the shaft; B, a front view showing perpendicular dotted lines, 1 to 10, which indicate the ten stamps of two batteries. The half-circumference of A (the cams being double) is divided into ten equal parts, commencing at 1. Through these points of division parallel longitudinal lines, 1, 2, 3 to 10, are drawn, and on the perpendicular line of each stem the dots *a*, *b*, *c*, *a'*, &c., are marked as follows, corresponding with the dots in the front view :—

First line, first cam, for the first battery = *a*
Second „ first „ „ second „ = *a'*
Third „ fifth „ first „ = *e*
Fourth „ fifth „ second „ = *e'*
Fifth „ third „ first „ = *c*
Sixth „ third „ second „ = *c'*
Seventh „ second „ first „ = *b*
Eighth „ second „ second „ = *b'*
Ninth „ fourth „ first „ = *d*
Tenth „ fourth „ second „ = *d'*

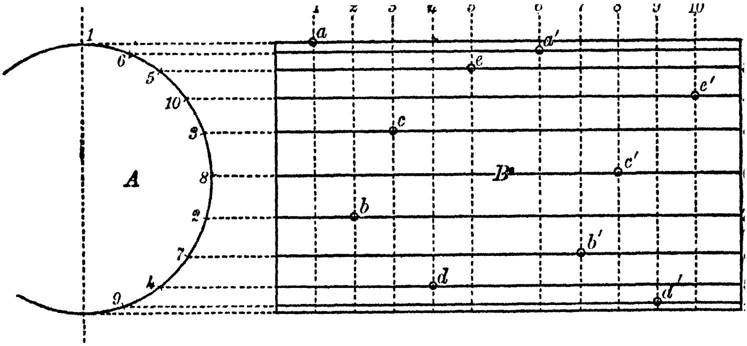

FIG. 20.—CAM SHAFT.

The points of division upon the circumference of A indicate the positions of the key-seats, which are transferred on to the cams, and seen in Fig. 21. The first key-seat on the first cam of the first battery, on the line *h*, is marked exactly in the same line on all of the ten cams. The distances from 1 to 2, then from 1 to 3, 4, and 5 (as seen in Fig. 20) are correspondingly transferred proportionately from 1 (on the line *h*) on to the second, third, fourth, and fifth cam of the first and similarly on to the following five cams of the second battery. When fastened on the shaft, and having a long common key-seat, the cams will stand as shown by the dots *a*, *b*, *c* in Fig. 20.

The Journal Boxes are constructed as shown in Fig. 19.

The journal cap is held down on the feed side by a wooden wedge, *a*, which is secured by a bolt passing through a slit, to allow, when necessary, of its being tightened.

The leading principle to observe in connection with lifts is, first, to distribute the weight of the stamps equally on the shaft; and, secondly, to have the next stamp on each side begin to rise before the middle stamp drops, so that the rock may be thrown on to both the right and the left dies.

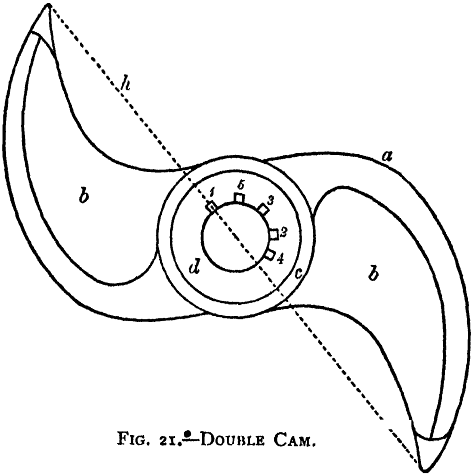

FIG. 21.—DOUBLE CAM.

Flange Coupling.—On the cam shaft is a bevel clutch or toothed coupling, *v*, Fig. 4, by which the cam shaft may, by means of a fork or bevel, be disengaged from the pulley shaft. It also fills the important function of preventing the breaking of the cams and the bending of the stems in case of a reverse motion of the cam shaft.

The Cams (Fig. 22).—These are of cast iron, having a face of 3 in. and a depth of 1 to 2 in. They are strengthened by a

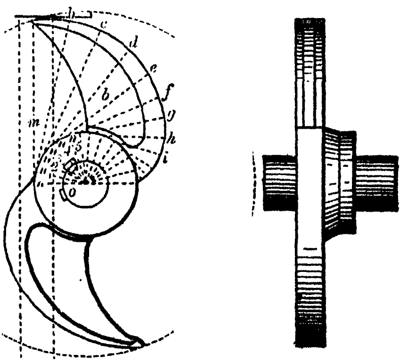

FIG. 22.—CAMS.—Scale ⅜ in. = 1 ft.

rib, *b*, 1¼ in. thick and increasing in depth towards the hub, which latter has a thickness of 2½ in. The hub is always on the off side of the stem, and therefore allows the cam shaft to closely approach the centre of the stem. The stamps are lifted by means of the cams. The cam face is generally constructed in the form of an involute curve, as by this construction the same point of the tongue or tappet is, during the lift of the stamp, raised vertically and uniformly, the lift of the stamp always having a proportion to the motion of circle described by the cam. Two cams placed opposite each other (Fig. 22) are always attached to one hub, which is strengthened by a wrought-iron band.

In California the cams for revolving the stamps are double. Single cams are in less frequent use, and their purpose is to increase the number of lifts, doubling the speed of the shaft, and, of course, necessitating additional power.

The construction of the cam is shown in Fig. 22. A *n* shows the distance from the centre of the cam shaft to the centre of the stem ; *n a* the length of the greatest lift, plus the distance from the centre of the cam shaft to the first front of contact of the cam with the tappet. The curve, *b a*, is changed to *b a′* ; *a a′* is in this case ⅜ in. The lift varies between 8 and 11 in., and is generally 10 in., with a corresponding cam curve of 21½ in. The friction between the cam and the tappet causes the stamp to revolve, thus insuring an even wear of the shoe.

The mortar (Fig. 13) is covered with 3 in. planks, *e*, held in position by the bolts, *m m*, which are attached to the side of the mortar. They join in the centre line of the stems, and are provided with semi-cylindrical grooves to receive the stems.

The Guides, which are of sugar-pine, are 15 in. deep, and 10 in. broad. They are firmly bolted to the uprights which carry the cam shaft. The centre of the upper guides is 3 ft. 4 in. above, and the centre of the lower guides 4 ft. below, the centre of the cam shaft. A wood lining, consisting of two 3 in. planks, 15 in. deep, with semi-cylindrical grooves to fit and enclose the stems, is bolted on to the main guide-timbers. The complete arrangement is shown at *s s*, in Fig. 4.

Discharge in Wet Crushing.—The discharge of the crushed ore takes place at one of the long sides of the mortar. The discharge generally takes place either through a slit or a longitudinal opening. For coarse and fine crushing—that is, the production of fine sand, not slime—the best arrangement is to use grates or screens. The discharge or sieve frame, which is made of cast iron, is divided into five panels (Fig. 14), each measuring 11½ in. by 15 in.

The Screen is of nearly the same size as the panel, only exceeding it by about ½ in. all the way round, and is bolted close to the panels by means of a cast-iron frame, 3 in. deep (Fig. 23). In order to make the joint tight, a blanket binding is sewn round the edge of the screen.

For the screens punched copper or iron plates are mostly used. Frequently brass or steel wire cloth is used, which produces a more effective discharge, as the surface of a wire cloth has more openings of the same size than the plates. The screens generally in use are made of Russian sheet iron (Fig. 24), weighing nearly a pound per square foot. The rectangular holes, which are punched by machinery, are ⅜ths of an inch long, and wide enough to allow the passage of a No. 6 sewing needle. The holes are ¾ths of an inch apart longitudinally, and their centres ⅝ths of an inch apart vertically, thus leaving a space of ¼th of an inch between them. The turned-up edge faces are on the inner side of the mortar, and when the edges are worn down the holes are closed or made smaller by pounding the edges with a mallet.

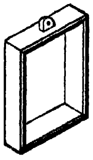

FIG. 23.—SCREEN FRAME.

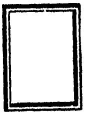

FIG. 24.—SCREEN.

Scale ⅜ in. = 1 ft.

To prevent choking, wooden hammers are sometimes connected with the shaft, which strike the screen at regular intervals. In the absence of this or some other arrangement answering the same purpose, the sieves must be brushed from time to time.

The Splash-box, which is of cast iron (Fig. 25), is provided with three discharge spouts, and is tightly bolted, with the aid of blanket packing, to the mortar.

Sieves in an inclined position give a better discharge than those placed perpendicularly.

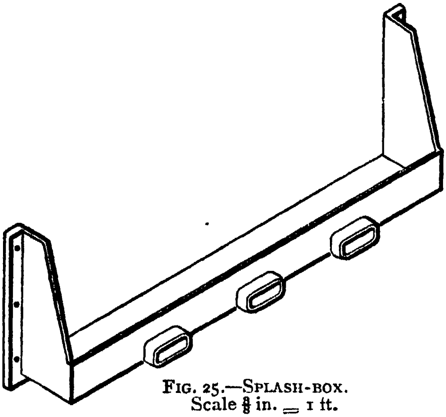

FIG. 25.—SPLASH-BOX.
Scale ⅜ in. = 1 ft.

In some of the mills of California and Nevada the discharge through the screens takes place in dry and wet crushing at the two long sides, but it is usual for the discharge to be on one side only.

The dies are placed from 2 to 3 in. below the discharge. There is very little difference in the method of crushing.

The manner of bracing the uprights of the cam shaft is seen in Figs. 5 and 10. As the strain of the belt is in the direction of the discharge, no bracing is needed on the feed side, which gives therefore a clear working space on that side of the battery.

The stamps are usually raised for repairing by means of a tackle, which is suspended on a roller strap on an overhead beam. They are kept in suspension by the props, *q* (Fig. 5).

The Supply of Water to the battery is effected by an iron pipe, *h* (Fig. 5), passing over the feed opening near the top of the mortar. The vertical discharge is through small apertures, which can be closed by wooden plugs. There is also a 1½ in. water pipe running along the discharge side of the battery, to allow for an increase of the quantity of water over the blanket sluices.

The quantity of water in the battery necessary to discharge the ore depends on the quality of the ore. Pure quartzose ore requires less water than clayey stuff. Coarse-grained ore or gold will bear more water than that in a fine condition. The quantity of floating metal and slime increases with a decrease in the quantity of water. The stamping of muddy or clayey ore sometimes requires more water than is necessary for a proper discharge, in which case water may be conveyed to the mass outside the battery. The settling of the particles of the ore is more perfect when the pulverised mass is in a sufficiently diluted condition. When crushing quartz rock in the usual way with a No. 4 screen, each stamp requires per minute from one-half to three-fourths of a cubic foot of water.

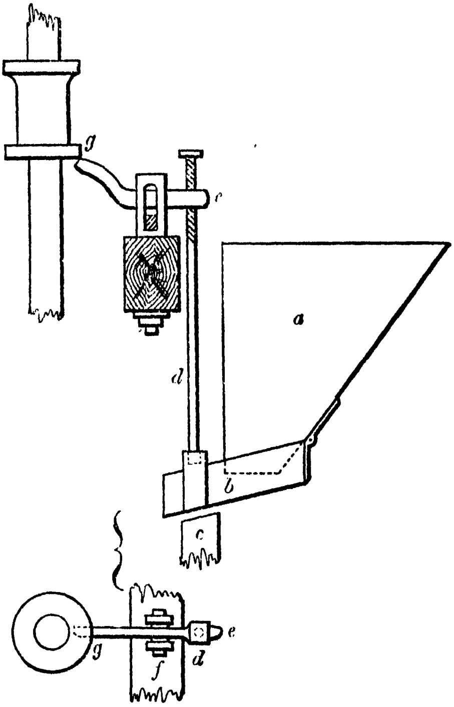

FIG. 26.—SELF-FEEDER.—Scale ¾ in = 1 ft.

Feeding the Battery with Ore.—This operation is performed either by hand or by some mechanical contrivance. Hand feeding is still in favour in California and Nevada, notwithstanding that a great number of mills are provided with self-feeders. The stamps always regulate the feed where self-feeders are employed. The one mostly in use is a movable feeder.

A Self-Feeder is represented in Fig. 26. The feed-box, *a*, of which there are generally two to each five-stamp battery, is fixed, and has a movable shoe, *b*, resting on the support.

The rod, *d*, is connected with the lever, *e*, which rests on a half-round pin, *f*. The space below *f* serves to regulate the position of the lever. It is filled with a wooden piece, placed higher or lower according as the wear of the shoe of the stamp may require. By the screw thread, *d*, the slide of the shoe, *b*, may be altered. The depth of the box, *a*, varies according to the quantity of rock it is required to contain. Small boxes are usually preferred, notwithstanding that there appears to be an advantage in having to fill a feed-hopper only once instead of perhaps four or six times a day. Fig. 27 shows Tulloch's ore-feeder, of similar construction.

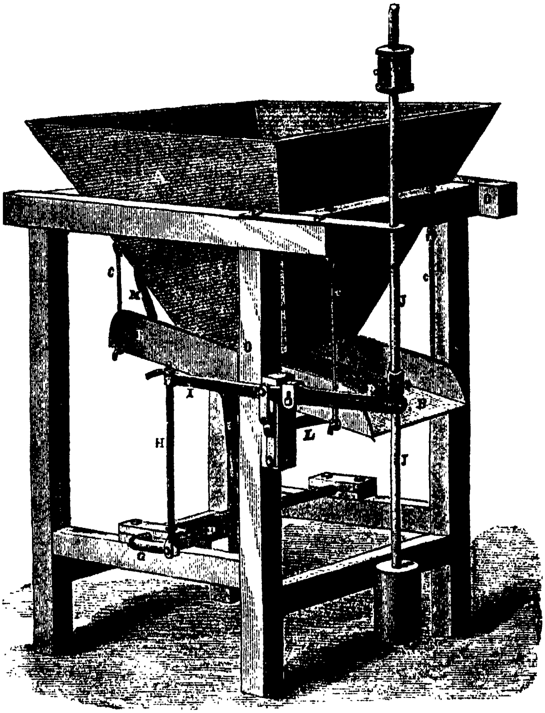

FIG. 27.—TULLOCH'S ORE-FEEDER.

Tulloch's Ore-Feeder.—In this machine (Fig. 27), A is the hopper into which the crushed ore from the stone breaker is discharged: B, the shaking tray for feeding the ore into the mortar-box; C, C, C, are suspension links for carrying the tray; D G is a rocking shaft which imparts a rocking motion to the feeding tray through the arm E; the lever I P is centred at L and is connected to the shaft D G by means of the link H; by means of the rod J, the top end of which comes in contact with the tappet of the centre stamp (a hole being bored through the lower guide block of stamps for this purpose), a motion corresponding to that of the stamps is given to the lever I P; O is the framework of the machine, and M is a strong steel spring for giving a sharp recoil to the tray B, thereby insuring a proper movement of the ore over the same.

These feeders are exceedingly simple and compact, and save all hand labour in feeding the battery. They can be regulated to feed any quality or quantity of ore. If the dies are deeply covered with ore the stamp does not fall through its full height, and consequently little motion is imparted to the shaking tray; if, on the other hand, there should be no covering of ore on the dies, the tappet will travel through the full height of its fall, and the rod J will move in proportion.

Proper feeding lies between these two extremes, and therefore the rod J leading to the lever I P should be so adjusted that there shall always be a thin layer of ore on the dies, and thus avoid any loss of power from unnecessarily pounding the ore in the mortar box.

II.—AMALGAMATION OF THE FREE GOLD.

The Attwood Amalgamator (Fig. 28) consists of two hollow cylindrical troughs, *t t*, 17 in. long and 4 to 5 in. deep, and made of wood or iron. These are filled with clean quicksilver, over which the blanket washings are directed. The gold, being heavier than the quicksilver, will sink to the bottom, except

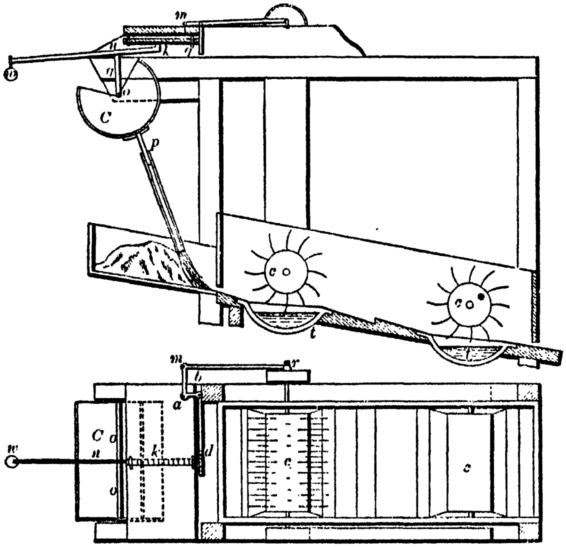

FIG. 28.—ATTWOOD AMALGAMATOR.

that part attached to the quartz or sulphuret, which is borne to the top.

The surface skimmings are agitated by wood cylinders, *c c*, 8 in. in diameter, suspended parallel to and over the centre line of the trough, and provided with radial arms of $\frac{1}{4}$-in. round iron, whose ends are slightly curved. These arms are placed along the cylinders in twelve longitudinal rows, consisting alternately of eight and nine arms, those of each row being set opposite the spaces in the next. They are not allowed to dip into the quicksilver, but they almost touch it. The cylinders are, between the centres, 2 ft. 10 in. apart, and are 6 in. below each other. They make sixty revolutions per minute, and are driven by small belts. The arrangement for insuring a steady flow of the sand from the blanket through the amalgamator is shown in the figure. The connecting-rod, *m r*, receiving a horizontal movement from the crank, *r s*, transfers it by means of the bent lever, *m a b*, to the ratchet, *b d*, which moves, by means of the wheel, *d*, the endless screw, *k*. The endless screw, *k*, gives a rotary motion to the discharge pipe, *p*, which is attached to the water trough, C, by means of a lever, *n*, connected by the rod, *j*, with the water trough shaft. By raising the weight, *w*, suspended at the end of the lever, *n*, the female screw at the opposite end of the lever, *n*, can be placed at any point along the endless screw, and a change in the inclination of the pipe may thereby be effected. The female screw on the lever, *n*, is part of a diametrically cut nut. On reaching the point, *g*, the endless screw ceases to move the lever, *n*.

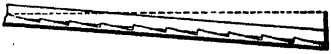

FIG. 29.—RIFFLE SLUICES.

One amalgamator treats the blanket washings of two 5-stamp batteries. The quantity of water passing through an amalgamator is one cubic foot per minute.

The tailings from the amalgamator pass through wooden riffle sluices (Fig. 29), which are generally two in number. They are each 9 in. wide, and have an inclination of 1 in. to the foot. The riffles are 6 in. apart, and at the lower end are from ⅝ in. to ¾ in. deep. They are filled with pure quicksilver and present a bright surface of from 2 to 3 in. in length by 9 in. in width. There are generally from twenty to thirty riffles to one sluice. The skimmings from the amalgamator and the quicksilver riffles, which vary from ·1 to ·5 per cent. of the crushed sands, are slowly ground and amalgamated in pans, which insures, with a minimum loss of quicksilver, a satisfactory result.

Hydrogen-Amalgam Process (*Molloy's Patent*).—The principle involved in this process is, that when gold is brought into absolute and maintained contact with clean or "quick mercury," the gold is absorbed by and retained in the mercury. As will be shown later on, where refractory ores have to be dealt with, they cause the mercury to "sicken"—that is, to become coated with an oxide which lies like a sheet of paper on the surface of the body of the mercury, preventing contact between the particles of gold and the clean portion of the mercury. This sick mercury also powders away—or, as it is termed, "flours"—so that the floured fluid metal is carried away and lost, leaving fresh surfaces to be attacked by the injurious ingredients in the ore, and causing a loss of mercury as well as of gold.

The object of the hydrogen-amalgam process is to save the loss of mercury and of gold by maintaining the "quickness" or "brightness" of the mercury, and by insuring a continual contact between each separate particle of the pulverized ore and the quick mercury.

The apparatus * consists of a shallow pan about one inch in depth, and 41½ inches in diameter, which contains mercury about half an inch in depth. In the centre of this pan is a porous jar, so placed and fixed that the mercury cannot enter or move it. Within this jar is a cylinder of lead and a solution of sulphate of soda. This lead cylinder, which constitutes the anode, is connected with the positive pole of a small dynamo machine, while the mercury is connected with the negative pole of the same dynamo.

When the current passes, oxygen is evolved from the surface of the lead anode, while hydrogen is evolved from the surface of the mercury. This action, which is apparent to the eye, is of course due to the decomposition of the electrolyte formed by the solution of the sulphate of soda. The mercury combines with a portion of the hydrogen, and so forms a hydrogen-amalgam, while the excess of hydrogen so formed passes away. Now, while the mercury is thus charged with hydrogen it cannot oxidize, because of the presence of an

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* I am indebted to the Hydrogen-Amalgam Company for the information here given concerning this apparatus.

excess of hydrogen. Thus, no matter what the character of

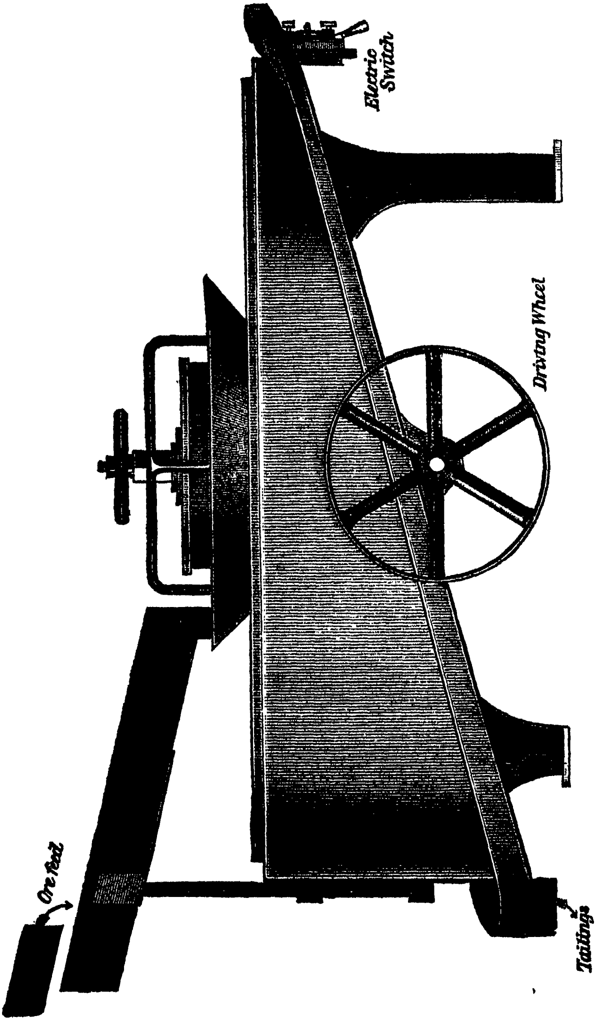

FIG. 30.—HYDROGEN AMALGAMATOR (MOLLOY'S PATENT).

the ore, the mercury, under these conditions, is always quick and therefore capable of attacking and absorbing the gold into itself.

For bringing the pulverized ore into contact with the mercury, the following method has been devised by the inventor. Floating upon the surface of the mercury is a disc forty inches in diameter, which dimension leaves a narrow outside channel all round the edge of the pan where the mercury is uncovered. The centre of the disc has a circular hole in it so as to clear the porous jar by about two inches. This central opening in the disc has a rim about two inches high which forms a hopper. The disc, as it floats on the mercury, is slowly revolved by simple mechanism. The pulverized ore, as it leaves the stamps or other crusher, flows into the hopper accompanied by a stream of water, and is then by centrifugal action carried under the revolving disc and rolled round in the mercury in ever-increasing circles until it reaches the periphery of the disc, and consequently the outward channel between the edge of the disc and that of the pan. Here, freed from the pressure of the disc, the pulverized ore floats up and over the edge of the pan and passes away, leaving behind it in the mercury every atom of amalgamable gold it previously contained. This perfect extraction is due to the rolling action which separates each particle of the ore and rolls it for some ten seconds in the bright quick mercury, which takes up in this way the *amalgamable gold.*

It is stated that the *increased* quantity of gold extracted by this process has never been less than 10 per cent., and that the whole cost of treatment amounts to only 3d. per ton for electrical and mechanical force and for labour.

Each amalgamator is said to be capable of treating from six to ten tons in twenty-four hours. The pulverized ore goes direct to the amalgamator from the battery. The maximum speed must not exceed 14 revolutions per minute, or mercury may be thrown out. It is recommended to crush very fine when using this machine—from 80 to 120 mesh per linear inch. The quantity of mercury for each amalgamator is 350 pounds.

Pan Amalgamation.—In many mills the heavy concentrates which accumulate on the blankets or other contrivances are submitted to amalgamation in grinding pans, of which several varieties are in general use. Two of the best known are the Knox and the Wheeler pans.

Grinding Pans.—(1.) *The Knox Pan.*—This pan is very much in use, and is capable of grinding, in three separate charges, the skimmings of a 30-stamp mill in twenty-four hours. Fig. 31 illustrates this pan in vertical section and plan. It is of cast iron, and is 4 ft. in diameter and 14 in. deep. The sides are ½ in. and the bottom ¾ in. thick. The pan is supported by four cast-iron legs, *l l*, bolted to the floor, and attached to the pan similar to a stove leg. On the legs are projections which support the cross-bars bearing the driving shaft. The vertical pinion-wheel can be put in or out of gear by a bevel clutch worked by lever.

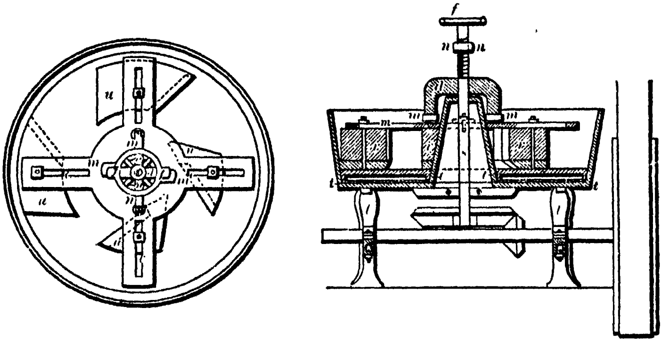

FIG. 31.—KNOX PAN.—Scale ¼ in. = 1 ft.

The pan has a false bottom, *t*, 1¼ in. thick, with a projecting vertical rim at the periphery, to form a hollow annular space underneath for the introduction of steam. There is also a radial groove in the false bottom, 1⅛ in. wide and 1 in. deep, for the accumulation of quicksilver and amalgam, which connects with the lower discharge hole situated opposite the driving shaft. The upper discharge hole is 4 in. above the lower one. The holes are closed by wooden plugs 2 in. in diameter.

The centre of the yoke, *d*, attached to the muller, *m*, is keyed to a vertical wrought-iron shaft, *s*, 2 in. in diameter, which latter is guided by a hollow cast-iron cone in the middle of the pan. The muller consists of a flat ring made of cast iron, having an inside diameter of 10 in. and being 4½ in. wide and 1 in. thick. Attached to the muller are four arms, 12 in. long and 6 in. wide, placed at right angles to each other, to which cast-iron shoes, *u*, 1½ in. thick, are bolted through the slits *c c*.

Between the muller and the shoe, a wooden shoe, *r*, having exactly the same shape as the iron one, and about 6 in. high, is introduced to prevent the unground pulp from settling on the latter, the upper face of the wooden shoe reaching above the surface of the pulp. The head of the bolt, passing through the shoe and muller arm, fits into a recess in the bottom of the iron shoe, and wears off gradually with it.

The yoke, *d*, bolted to the muller, serves the purpose of raising or lowering the latter by means of the screw, *f*, which rests on the vertical driving shaft and passes through the centre of the pan.

The muller makes from twelve to fourteen revolutions per minute, the force for which is communicated from the horizontal shaft by bevel-gearing which makes from thirty to thirty-five revolutions per minute.

Process of Pan Amalgamation.—This is effected by charging 300 lbs. of the skimmings or blanket washings into the pan, and adding water until the pulp just adheres to a stick dipped into it without dropping off. Then the pulp, after three hours' grinding, is heated by steam under the false bottom of the pan. At the same time is added a cupful of a mixture consisting of equal parts of saltpetre and sal-ammoniac. About 50 lbs. of mercury for every charge is also added simultaneously with this mixture.

After being amalgamated for three hours, the pulp, then very fine, is diluted with water. A few handfuls of caustic lime are added at the same time, which greatly assists the coagulation of the particles of quicksilver. The diluted pulp, which reaches within one or two inches of the top of the pan, is agitated for about twenty minutes, after which it is discharged, the muller being kept in motion, through the upper plug hole, and, subsequently, through the lower. A bucket, which is placed in front of the discharge holes, catches any quicksilver or amalgam which escapes during the discharge. When a clean up is required, the quicksilver and amalgam and the small quantity of sulphides still remaining are washed into the bucket. Quicksilver is then added and the skimmings removed,

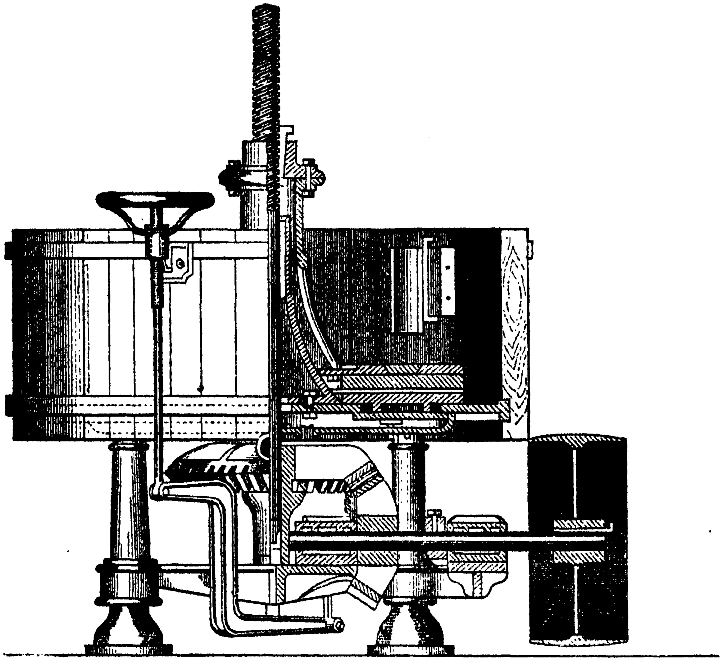

FIG. 32.—WHEELER PAN WITH WOODEN SIDES.

which go back again to be re-ground. The amalgam is worked by hand and the lumps broken up. The impurities on the surface are removed and the bright quicksilver strained through a canvas filter.

The pulp from the pan is run into a large tank to settle, and in many cases it still retains a considerable quantity of gold, amounting sometimes to a value of £25 per ton, which is added to the coarser sulphurets and treated by a process to be hereafter described.

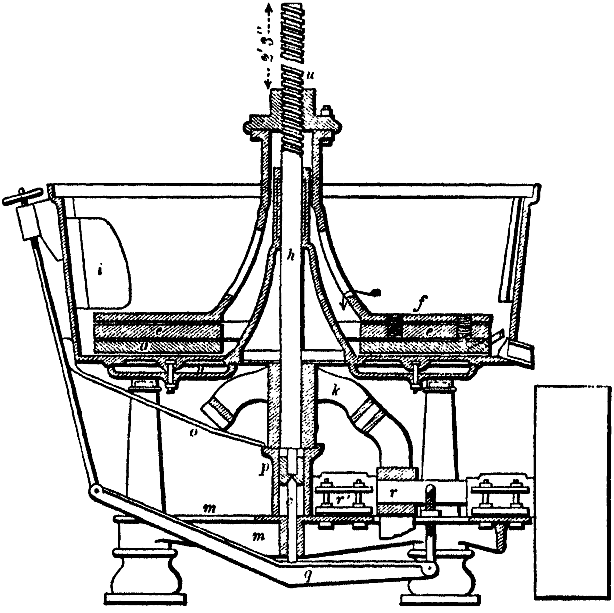

Fig. 33.—Wheeler Pan.—Scale ½ in. = 1 ft.

(2.) *The Wheeler Pan.*—Figs. 33 and 34 show one of these improved pans. Fig. 33 is a cross section on the line 2 of the ground view, Fig. 34, one part of which exposes the dies, and the other part the muller with the shoes, shown by dotted lines.

The pans have a cross piece, *m*, in which the box, *r'*, for the horizontal shaft, *r*, the step box, *p*, and the lever, *q*, are fixed, which vary only in shape and arrangement. The lever, if screwed up by the hand wheel, lifts, by means of the pin, *v*, the box, and with it the perpendicular shaft and muller. The upright shaft, *h*, is provided with a screw, *u*, by which the muller is screwed up so high above the rim of the pan that cleaning and changing of the dies may be conveniently performed. By this arrangement the hoisting apparatus is avoided. The guide plate, *i*, or, as it is called, the scraper, of which there are four in a pan, is short, which gives to the moving pulp sufficient direction towards the centre, and may be readily removed from the T-shaped projection of the pan; *o* is a pipe which conveys oil to the box on which the hub of the gear, *k*, runs; *n* shows the steam chamber for heating the pulp. The muller, *f*, is provided with two openings, *b b*, to receive the dovetailed projections of each shoe, as shown in Fig. 35.

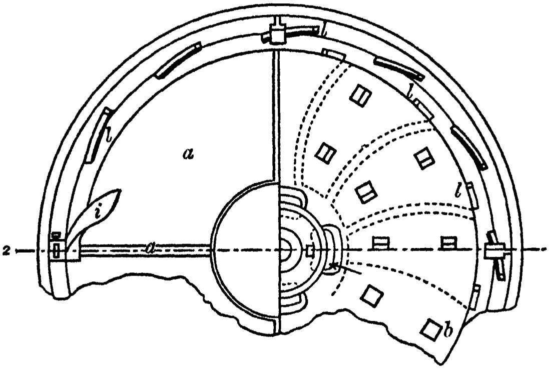

Fig. 34.—Ground View of Wheeler Pan.—Scale ⅛ in. = 1 ft.

There is a very simple method of fastening the shoe, *e*, to the muller. *c* is a vertical section on the line *a*; the shoe is placed under the muller, so that the projections, *b b*, fall into the openings in the muller; the space, *k*, is then filled by a wood wedge, when the shoe stands firm. The dies, *a*, Figs. 33 and 34, of which there are four to each pan, have a similar dovetailed knob, which is wedged into a recess in the bottom of the pan.

In order to impart to the stuff a rising current on the periphery of the bottom, there are on the sides of the pan inclined ledges, *l*. Smaller ledges, *l′*, for the same purpose, occur on the periphery of the muller, but these latter are inversely inclined. The space, *a′*, between the dies can be wedged with hard wood. Each pan is capable of a charge of 600 to 800 lbs. of slimes.

It is evident that a pan having a circular plane bottom must increase the grinding from the centre towards the periphery, as each point of the muller on the radial line describes a larger circle proportionately as it is farther from the centre, and must therefore wear accordingly. If the muller on this account wears sooner at the periphery, a partial suspension of the work appears to result, the points near the circumference waiting as it were until the wear towards the centre overtakes that at the periphery. It would seem, therefore, that if the muller and the dies could be constructed of a material continually increasing in hardness towards the periphery, the grinding and the wear would be rendered uniform.

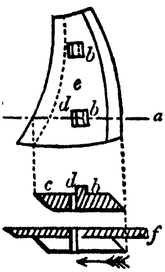

FIG. 35.—THE SHOE. Scale $\frac{1}{8}$ in. = 1 ft.

It is also evident that the quantity of quartz which enters the central opening must decrease proportionately as it approaches the periphery; that is, considered in reference to the increasing grinding surface of the muller. The quartz is also being ground while on its way towards the periphery, which receives six times less quartz than the central opening (considered in reference to the larger circle), the quartz being already partially ground. It is obvious, therefore, that the wear of the muller has for this reason a tendency to decrease towards the periphery. It is necessary also to consider the radial slots between the shoes of the muller, which supply the grinding surface farthest from the centre with fresh stuff; but as the sectional size of the slots decrease by the wear of the shoes and also by their being choked with amalgam and other matter, this evil tends somewhat to rectify itself.

(3.) *The Improved Soderling Pan.* — Mr. Soderling, a Californian mining engineer, has recently introduced a form of pan which possesses novel features, and will no doubt prove

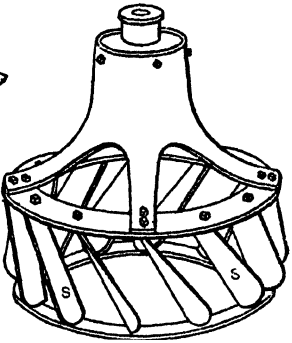

FIG. 37.—STIRRING APPARATUS

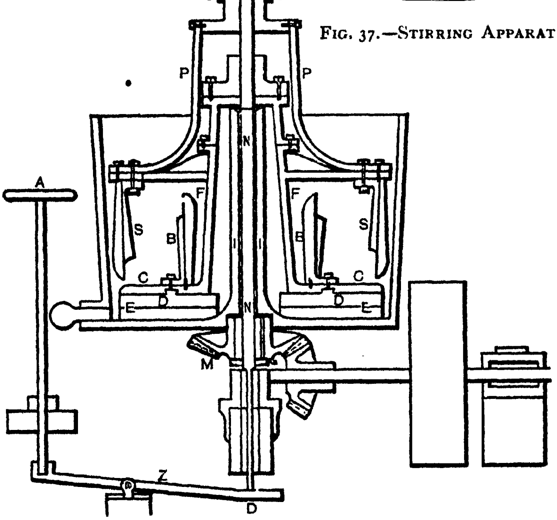

FIG. 36.—SODERLING'S IMPROVED AMALGAMATING PAN.

of great advantage. It should prove especially valuable in the treatment of slimes and tailings which, through long exposure, have become baked together, as the disintegration of lumps can

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be effected in this pan without any previous mechanical preparation. In the pans described on the previous pages, the material is caused to pass between the mullers and dies, and the two oppressing surfaces grind and reduce the pulp in the presence of mercury, to save the precious metals. In this pan the material itself acts as a grinder, by the production of a counter-current. Its construction will be seen from the accompanying illustration (Fig. 36).

The pan can have a steam chamber at the bottom, and is provided with the muller (C) to which the grinding shoes (D) are attached. Dies (E) are placed in the bottom of the pan, and between them and the shoes of the muller the material is caused to pass by the centrifugal motion of the muller from the inner portion of the pan to the outer. This muller has a yoke (F) or cone extending upwards above the central cone of the pan, and is driven by means of a hollow shaft (N N), which extends up through the central cone (I I) of the pan and has a feather or key by which it is connected with the muller. This hollow shaft is driven by means of a bevelled gear and pinion M. A shaft extends up through the hollow shaft, and has a gear wheel (Z) fixed upon the opposite side from the other gear wheel, this one pinion serving to drive the two shafts in opposite directions.

Another driving arrangement, where the bevel gearings are arranged one below the other, is shown in Fig. 38.

The upper end of the solid shaft has a feather which engages with a corresponding keyway in the hub O at the upper end of the outside cone P P. This cone extends down outside the inner one F, which connects with the muller corresponding with it in shape, and is connected with a stirring apparatus which is driven by it in an opposite direction from that of the muller. This stirring apparatus S S (shown in Fig. 37) comprises rings having fitted thereto angular blades, S S, Fig. 37, thereby forming slots between the blades, these slots being angular, and having their sides curved, so that when rotating the tendency will be to carry the material from the outside of the pan through the slots and throw it toward the centre, where it will come in contact with the oppositely-moving current, which is thrown outward by the inner muller moving in the opposite direction. This action produced by the independent currents forms a circulating current above and below the mullers, and causes the harder and softer portions composing the pulp or slime to erode and disintegrate each other, and the whole mass of material will be acted upon by the chemicals employed during the process, and will be amalgamated with the mercury which is placed in the pan for the purpose.

This counter current can be increased by attaching to the muller the blades B B, which are fastened to them in the manner shown in Fig. 36. The pan, the steam bottom, dies, the muller and driving gear, are similar to those used in the ordinary grinding pans; and (as shown in Fig. 37) the extra or circumferential outer muller can be attached to any pan now in use at a small cost and be rotated from above, by a shaft, the lower end of which runs in a deep slip or guide on top of the ordinary muller, and at the upper end feathers to engage with corresponding key-ways cut in the side, the hole that fits the shaft in a gear wheel (or pulley to apply power direct by belt) in either case with a long hub, running in a boxing. The rotation of the outer or circumferential muller produces an action much greater than can be accomplished by grinding, the time being no greater than that occupied with the ordinary grinding.

The amount of power required is only that necessary to run this isolated ring, which may be made to run either faster or slower than the grinding muller by a simple change in the relative sizes of the gear wheels. By this device friction is avoided, which would be necessary if more grinding surfaces were employed. The excessive wear of the machinery is also avoided, and the action is better, inasmuch as the striking points of the opposing currents are acting all the way round the pan, and so far from the centre as to be in operation about one third the radius of the pan from the edge, showing the impact action to increase in ratio to the cubic contents, of great importance in building large pans.

For treating tailings, pans of this construction can be utilized holding 5-ton charges, running four hours, which will give a daily capacity of 30 tons per pan, and five pans could reduce 150 tons. According to the coarseness or fineness of the tailings, grinding with the muller and the shoes will be only required for an hour or so.

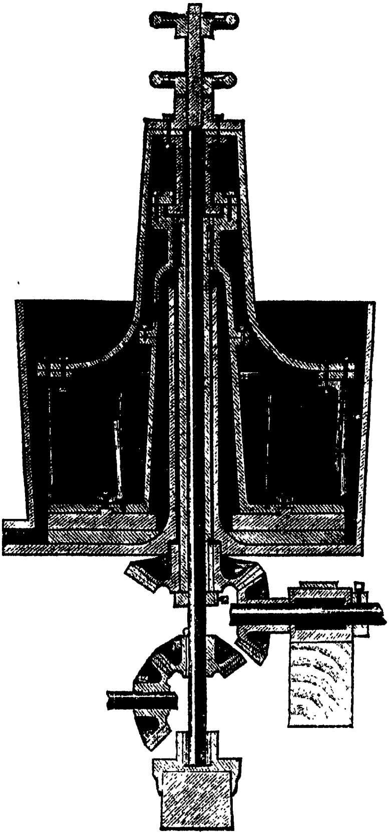

FIG. 38.—SODERLING PAN.

When it is desired to clean up, attach chain block and hoist; and the shaft being small below the hub will slide up through the muller up against the hub, 7 to 10 feet above the pan, in this way giving ample room to remove the ordinary muller and dies as usual. The lever Q, worked by hand-wheel A, is intended to raise or lower the muller. The shaft is connected by feather to the gear-wheel, which has a groove in the hub for a support to hold it in its place under the pan bottom.

The Settler.—After several hours' grinding in the pan, the pulp is discharged into another vessel similar to a pan, which is called a "settler" (see Fig. 39). This consists of a circular vat, in which revolves a central axis, to which iron shoes are attached. The shoes are not to come in contact with the bottom, but are faced with wooden rubbers, so as to keep the heavier parts of the pulp thoroughly stirred. Water is introduced during the operation, and the pulp can then be drawn off at plugged holes as required. An iron bowl for con-

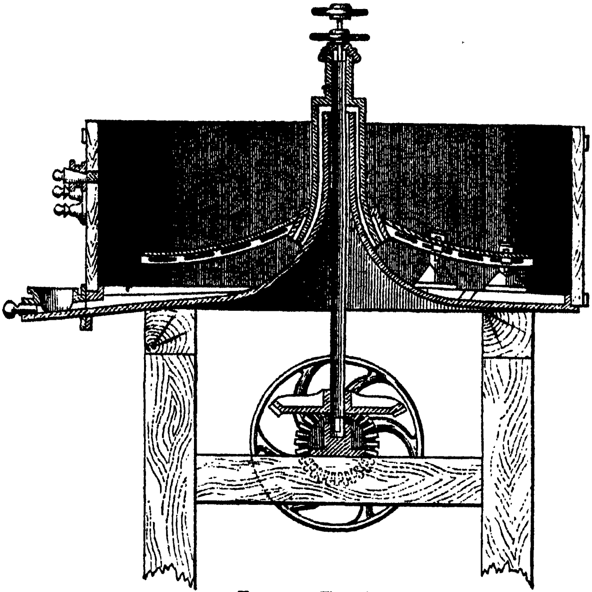

FIG. 39.—THE SETTLER.

taining the quicksilver is attached, which communicates with a radial gutter cast in the bottom of the settler.

Some mills are differently arranged, and the sands, after leaving the blankets and the amalgamators, discharge themselves into the rubbers, whereas in other mills the pulp, as soon as discharged from the copper plates, passes into the rubbers; but these machines are being rapidly replaced by better and more modern amalgamating appliances.

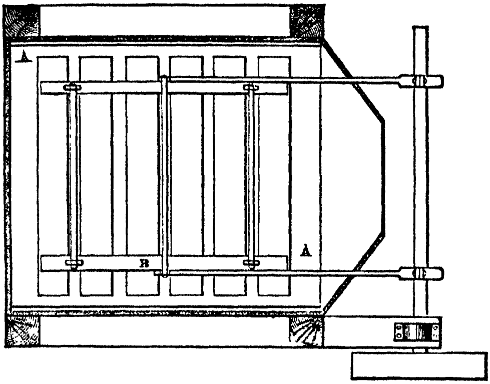

FIG. 40.—EUREKA RUBBER.—Scale $\frac{5}{16}$ in. = 1 ft.

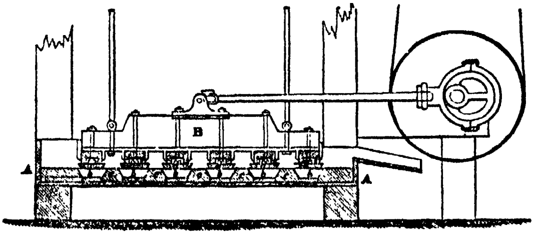

FIG. 41.—EUREKA RUBBER.—Scale $\frac{5}{16}$ in. = 1 ft.

Rubbers.—In the rubbers the particles of gold are further cleaned and brightened, while at the same time being detached from the sands. They have also an opportunity of being caught by the amalgamated copper plates of the rubber.

The Eureka Rubber (Figs. 40 and 41) consists of a rectangular cast-iron box, 7 in. deep and 4 ft. 8 in. square, which is provided with a false bottom made of cast-iron dies or plates. On these latter are cast-iron shoes affixed to a wooden frame, which have a rectilinear motion imparted to them by rods connected with an eccentric. The wooden boards for the shoes are covered with amalgamated copper plates.

In preparing the false bottom of the rubber box, the boards *ccc* (Figs. 40 and 41), which are 3½ in. wide and 1 in. thick, and of the same length as the box, are laid in at right angles to the motion of the frame, and have a space of 4 in. between them. On top of these boards are placed, broadside upwards, the plates or dies, *b b*, which are 1½ in. in thickness. Between the edges of the plates a space of 3 in. is left. The spaces, *c c*, are filled with pieces of wood, which are set up endwise, and dressed to fit the false bottom plates. They are 2½ in. long, 4 in. wide, and from 6 to 12 in. broad. For rubbing, this arrangement is found to answer better than a false bottom made of solid cast iron. The lower boards, *e e*, are introduced to give a firmer fixture to the blocks, *c c*, when the iron false bottom plates, *b*, have become partly worn. The plates are firmly secured by strips of wood, *g g*, 1 in. thick, bolted to the sides of the rubber box, and fitting into the recesses, *f f*, on both ends of the bottom plates.

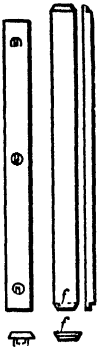

FIG. 42.
FALSE BOTTOM.

In Fig. 44 is shown the construction of the movable rubber frame. Both the iron and the wooden shoes, K K, which carry the amalgamated copper plates, fastened by iron clamps and bolts, are secured by two cast-iron frames, *l l*. Each of these frames is attached to a piece of timber, *g*, as shown in the figure. The movable rubber frame is

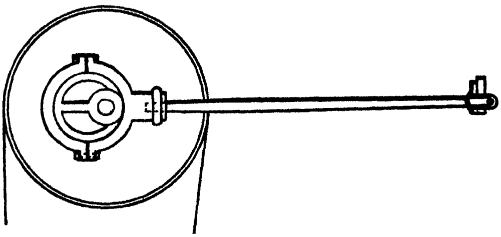

FIG. 43.—ECCENTRIC.

attached by four wrought-iron rods, *a a* (Fig. 41) to a wooden frame, *m m*, which can be raised or lowered by means of two screws, supported on the main wooden frame. The main frame, which supports the rubber box and the movable rubber frame, is constructed of pine lumber 6 in. square. The ends of the bolt, which passes through the journals, *j*, form the pins to which the eccentric rod (Fig. 41) is attached. The number of revolutions of the eccentric shaft is 55 per minute, and the length of the stroke of the rubber frame is 4 in. There is one rubber to every battery of 4 or 5 stamps.

FIG. 44.—MOVABLE FRAME.

The sands after passing the rubber flow over the amalgamated copper plates. These plates are 2 ft. 6 in. wide, and are made of soft copper and heavy material; they are laid in wooden sluices, which have an inclination of ½ in. to the foot. The sides of the latter are 3 in. high, and their length is from 20 to 30 ft. Three inches per stamp is the usual width of the copper plate sluices.

Amalgamated Copper Plates.—Of course, the amalgamated copper plates only catch the gold which is in a free state. The principal difficulty with which the millman has to contend is the discoloration of the surface of the plates by oxides and insoluble metallic salts, as it is necessary to keep the surface of the plates bright and free from film, which would prevent an easy and sure adhesion of the particles of gold and amalgam as the pulp passes over. A new plate is not considered in fit condition until it is capable of keeping bright without dressing for several hours. This condition is not usually attained until a firmly adhering layer of gold amalgam has attached itself to the plate, and this does not generally take place till after the loss of much gold, labour, and time. Silver-plated and even gold-plated copper plates have been tried, but they have not met with much favour, as they have proved either ineffectual or too expensive.

In the preparation and care of amalgamated copper plates, in order that they may be put and kept in the most effective condition, and at the smallest expense, regard must be had (1) to the quality of the copper; (2) to the method of amalgamating the plate and of dressing it; (3) to the purity of the mercury and of the water; (4) to the character of the ore. The plate should be of the purest and softest copper, and free from dark, rough spots. The kind known to the trade as "braziers' copper" is the best material for selection. That which has been rolled with special care in order to make the surface smooth and hard will not be suitable, as a plate made from it would not absorb the quicksilver so well as the softer and therefore more porous copper. It is best, from the point of view of efficiency and durability, to select plates weighing not less than 3 lbs. per square foot. It would be better even if they were thicker than this. For inside plates, copper two or three times as heavy as this should be used, so that the plates may not be easily bent or torn from their position. If possible, copper which has been annealed subsequently to the last rolling should be selected. If this cannot be had, then plates annealed by exposure to heat on their under side, sufficiently to ignite sawdust laid upon the upper side, will be the next best. This may be done over a blacksmith's fire, but more conveniently over an open wood or charcoal fire. Every part of the plate should be subjected to the heat. This process will soften the plate, and make it more porous; it will therefore be capable of retaining more quicksilver and amalgam than it otherwise would. As the efficiency of copper plates is in proportion to their ability to hold quicksilver, this point should receive careful attention.

The plate is straightened by laying it on a table and using a wooden block and hammer. The hammer should not be struck directly on to the plate, but the block interposed so that the face of the copper may be compressed or drawn into shape. The plate should then be fastened to the table with iron screws —or, preferably, copper nails—long enough to clinch on the under side of the table. In either case, the heads of the screws or nails should be flush with the face of the plate. Brass screws should not be used, as the quicksilver soon penetrates the heads and makes them brittle, and thus destroys their capacity to hold. The minor inequalities of the surface of the plate may then be removed by the block and hammer.

The plate should next be scoured with wood ashes and fine sand or tailings applied with a scrubbing brush or coarse rag. In case of bad spots, the scouring should be done by means of the end of a small block of wood. The scouring must be continued until the coating or oxide is entirely removed, and the bright, metallic copper exposed. Caustic soda, concentrated lye, or sal soda, may be used instead of ashes, to remove traces of oil or grease resulting from the action of alkalies. After washing with clean water, a solution of cyanide of potassium—say, half-an-ounce of cyanide to a pint of water—should be applied with a soft brush or swab.

To amalgamate the plate, a mixture of fine sand or tailings and powdered sal ammoniac, in equal parts, with a small quantity of quicksilver sprinkled in, may be used.

This mixture is applied with the scrubbing brush, and the scrubbing continued until all parts of the plate are amalgamated. As much quicksilver as the plate will absorb should be sprinkled on to the plate during the operation, with water enough to make a thick mud of the mixture. I have used lime with the mixture, but I cannot say that the addition gives any special value to it. It is certainly unpleasant, inasmuch as ammonia is set free in great quantities. The mixture must remain on the plate for an hour or so, and then be washed off by means of clean water and a brush, which should be followed by a soft brush and cyanide solution, adding quicksilver if the plate will hold it. By continuing this treatment for three or four rounds, the plate will be found to have taken up as much quicksilver as it would ordinarily after many weeks' running. If fine gold amalgam can be spared, it is well at this stage to rub some upon the plate, using a cloth rubber wetted with a solution of sal ammoniac, in the proportion of four ounces to a pint of water. If the amalgam be heated and rubbed in a Wedgewood mortar, so as to insure the solution of the gold, it will be found to attach itself more readily to the plate.

Silver-plated Copper Plates.—These plates (which I have used with excellent results) are in favour with some mill-men, especially when starting a new mill, as the silver amalgam will have more affinity for the gold than a copper plate simply coated with mercury; but there is nothing better than old plates, when once thoroughly "soaked" with gold.

The silver amalgam may be made as follows: Dissolve a piece of silver—coin will answer the purpose—in the smallest quantity possible of dilute nitric acid. Heat will be found to hasten the solution. Dissolve the resulting crystals of nitrate of silver in water, and pour into the vessel enough quicksilver to reduce and amalgamate the silver in the solution. This process is completed in a few hours. Wash the amalgam with clean water to remove any traces of nitrate of mercury, and then strain it to remove any surplus quicksilver. This will leave in the cloth a pasty amalgam of very pure and finely divided silver, which should be used as directed for the gold amalgam.

The rationale of this treatment may be stated as follows:—The first scouring removes oxide and grease, and exposes the bright metal, which by annealing has been, by virtue of its porosity, rendered capable of holding a comparatively large quantity of quicksilver and amalgam. The sal ammoniac, through dissolving the oxides, assists the amalgamation by preserving a bright surface to the plate, and thus allowing the quicksilver and amalgam to penetrate the copper, filling the minutest interstices, and combining with it in atomic proportions. The result is the production of an alloy of copper, gold, and quicksilver. The air, water, and the various salts being thus kept from acting directly upon the copper, and formation of oxide and carbonate of copper is avoided. s is accomplished in a longer or shorter period which ends upon the mode of preparation and the richness of the and also the ease with which the gold in the ore may be amalgamated. It is probable that the galvanic current arising from the contact of the two dissimilar metals plays an important part in the chemical reactions and the amalgamation of the gold, but its real action is not definitely known.

The main point in the work of preparation is to effect a quick union, in proper quantities, of the gold and the quicksilver with the copper. If this union takes place slowly, loss of gold, which is unavoidable, makes the expense much greater than would be the case if the process were assisted by a needful amount of amalgam, before running the ore over the plate. If a section of an old well-used plate be examined, the amalgam will be found to have penetrated perhaps half-way, and in some cases even completely through the copper.

After scouring the plate well with some sandstone, and subsequently rubbing it with a solution of sulphate of copper, the copper will also be in a condition to take up the quicksilver.

Corrosive sublimate, nitrate of mercury, sulphuric acid, and common salt, muriatic acid, sodium amalgam, cyanide of potassium, and other acids and salts, have all been used in amalgamating copper; but though amalgamation may be accomplished by them in some cases quite easily, yet the plate cannot be brought by them to a good working condition so soon, even if at all, as by the method I have described.

The subsequent treatment of the plates, amalgamated in the manner described, should be varied to suit the character and quality of ore and the purity of the water. Water containing carbonic acid discolours the plate, forming an insoluble coating of carbonate of copper. Though this coating is but a very thin film, it is nevertheless sufficient to prevent the adhesion of the gold and amalgam, unless these latter be in particles large enough to break through the coating. worst form of discoloration arises from the action of of iron and copper present in the ores, arising from decomposition of the pyrites. The addition of lime to water in the battery, as much as will dissolve therein, will found to neutralise the carbonic acid and decompose the phates. It may be necessary to add lime to the ore, so as to furnish sufficient to react on all the sulphates.

Dressing the Plates.—When the mill is in operation, the plates should be dressed every six hours, or oftener, if they are discoloured. After washing them with a stream of clear water, a solution of sal ammoniac should be applied with a small brush. The sal ammoniac should be allowed to remain on the plate for a few minutes, and the plate afterwards washed with clear water. Then, with a brush, enough of the solution of cyanide of potassium must be applied to brighten the plate. The plate should have upon it as much quicksilver as it will hold, which should not gather in drops or run off.

The quicksilver should be entirely free from base metals, such as lead, zinc, tin, and copper. Of course, the presence of gold and silver is desirable.

Chemicals.—The following are the chemicals it is necessary to keep at the mill:—Fused cyanide of potassium; sal ammoniac; caustic, or unslacked lime; caustic soda or concentrated lye; and nitric acid.

III.—THE PROCESS OF AMALGAMATION.

Operation of the Mill.—The stamping process is not so simple as would appear at first sight. The form of mortar, the mode of exit for the stuff, the weight and rapidity of the pestles, and the quantity of water employed, must be varied to suit the manner in which the particles of gold are distributed through the mass, as well as the structure and character of the ore and of the matrix. Fine reduction is by no means always desirable, for if some kinds of stuff be reduced too fine, much of the ore contained in it will be wasted. Considerable judgment is therefore necessary in selecting the screen best suited to the material which has to be reduced.

The rock is delivered from the mine in cars, each containing, say, 13 cubic feet, and weighing 1,300 lbs., in front of the rock-breaker. At custom mills the quartz is delivered in waggons, the crushing being paid for by the load, which consists of about 35 cubic feet, and weighs, including moisture, 3,500 lbs.

As a rule heavy stamps are more effective in the quantity of ore they crush than light ones. The stamps and the lift must both bear some relation to the hardness or the softness of the ore. Other circumstances, however, have also to be taken into consideration. In California, especially, where a hard, flinty quartz occurs, there are many mills with from 800 to 900 pound stamps, and with a lift of from twelve to fifteen inches. For fine crushing a high lift light stamp (say from 600 to 700 pounds) and a deep mortar are used. For coarse crushing it is found better to have heavier stamps, less lift, a shallow mortar, and more speed.

The capacity of a stamp mill is calculated by multiplying the weight of the stamp with the fall in feet and with the number of drops per minute, giving as the result in force per minute exactly the number of foot pounds exerted by each stamp. Dividing by 33,000—the number of foot pounds per minute in one horse-power—we have the horse-power per stamp, from which the effective power of the whole mill may be obtained. Dividing the amount of rock crushed daily by the effective horse-power will give us the daily amount per horse-power; and this is the best measure that can be obtained for the effectiveness of the stamps.

The object of crushing being to liberate the fine particles of gold contained in the quartz, so that they may be collected by the copper plates and blankets outside the battery for amalgamation, it is desirable that the crushing be fine. In this process quick crushing does not interfere with the saving of the gold, as it is collected and amalgamated outside the battery after crushing. It is this feature which distinguishes the blanket process from that of amalgamation in the battery during crushing.

The feeding of the battery should be so regulated that there shall not be at the end of any drop more than two inches of sand between the die and the shoe. A good feeder generally knows the requirement of the battery by the clear or dull sound produced by the stamp stroke.

A battery of 20 stamps weighing 850 lbs. per stamp, with 61 drops of 10 inches per minute, crushes, without the aid of a rock-breaker, 40 tons of quartz in twenty-four hours; while a battery of 20 stamps, weighing 700 lbs. per stamp, with 68 drops of 10 inches per minute, crushes 32 tons of the same rock; a No. 6 screen being used in both cases.

The power necessary to execute the work of the heavier as compared with the lighter stamps is as 850 × 61 : 700 × 68, and the work expected from them would be 35 tons for the heavier and 32 for the lighter stamps. The former, however, crush 40 tons, which is an additional quantity of over 5 tons in favour of the heavy stamps. When the rock-breaker is used, the proportional result is almost the same. The limit of weight has never been experimentally determined, though stamps are satisfactorily employed weighing over 900 lbs., and having a drop of ten inches. These results are on rock of the average hardness.

The quantity of rock crushed depends, of course, very much on the nature of the ore, the weight of stamp, the speed, the screens, &c. A 700-pound stamp will crush, at a speed of seventy-five blows per minute, using No. 4 or No. 5 screens, from one to three tons in twenty-four hours.

A slow motion of the stamp will produce more floating stuff and will crush finer than a rapid motion, because, in the case of a quick falling of the stamps, the small particles of ore are not allowed sufficient time to fall under the shoe, and are therefore carried out; while with a slow motion many grains which could pass the screen slide under the stamp, and are there pulverised finer than required. Although it is evident that a high speed will crush more stuff in a given time, yet there is a limit which the speed should never exceed. The stamp should fall on the ore with its full force, and the revolving stamp should have time to finish its motion. A quick or sudden blow is less effective, while at the same time there is the danger of a tappet being caught by a cam. The speed is excessive if it exceeds seventy-five blows per minute.

The rate of speed should be regulated according to the character of the ore. There are instances where, upon an increase of speed, the yield of gold per ton fell off; and practical trials alone, therefore, should determine the rate of speed in each mill. The rapid running of the stamps, and consequent augmentation of product crushed, causes greater agitation within the battery box, and requires a larger supply of water to clear the discharge and carry away the increased amount of pulp. The excess of agitation in the battery may prevent the accumulation of gold on the interior plates, and the excess of current on the aprons may prevent the accumulation of gold there. These objections are most plausible when the gold is most finely divided in the quartz.

Professor Raymond has the following observations on this subject:* "The conditions most favourable to economical crushing must be partly sacrificed to secure efficient amalgamation. The attempt to catch the greater part of the gold on the interior plates interferes directly with the efficiency of the stamps. The success of the amalgamation at this point is in inverse proportion to the success of the crushing and discharge. There is a certain advantage gained in the force with which the pulp is dashed against the plates; but this force is liable to overdo, and thus undo, its own work, and actually remove the adhering amalgam. The same effect can be more completely secured outside of the battery."

My aim in crushing has always been to collect as much gold inside the mortar as possible, that being the best place to collect and keep it safely, even at the sacrifice of crushing less rock per day; and I have always endeavoured to get the highest percentage of assay from the rock rather than to pulverise as much rock as possible, or to make up the average daily production by an increase in the quantity crushed. The conditions for collecting the gold when once outside the mortar are not as favourable as they are inside the mortar, as the pulp is swept over a small, steep, and smooth surface of amalgamated copper plates; and even when arranged in steps or drops, the chances are that a large percentage of the gold will be swept away.

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* Official Report on Mines and Mining.

In another passage, Mr. Raymond sums up his views as follows: "(1.) The stamp-mill is the most convenient and practically efficient machine for crushing quartz thus far introduced and proved by experience. It involves little waste of power in gearing; it delivers its power in the most direct and practical manner, namely, by blows, which take advantage of the brittleness of the rock, instead of pressure or friction, which invites the resistance of hardness; its capacities for charging and discharging are ample and easily regulated, both as to quantity and as to fineness of the product; it is subject to few and comparatively inexpensive repairs, and it can be repaired, in most cases, without complete stoppage. These and other excellent features in its construction and operation render it especially suitable for use in mining districts remote from machine shops, foundries, and centres of skilled labour.

"(2.) To obtain the best results, stamp-batteries should be built and run to secure the highest efficiency and economy in crushing only, without reference to amalgamation. The amalgamating apparatus should be adapted to the batteries, not the latter to the former. If interior plates are employed they should not be expected to catch the greater part of the gold, nor should the pulp escaping through the screens be swiftly and carelessly manipulated, when a little extra space and time devoted to it, almost without extra labour, would avoid much loss.

"(3.) The efficiency of the stamp may be described as the product of three factors—weight, fall, and speed. The efficiency of a battery of stamps involves a co-efficient—the discharge.

"(4.) When the fineness of crushing is regulated by screens, the discharge should be as large as practicable. There may be mechanical objections to continuous screens running around the whole battery; but there are, I think, no valid arguments against the double discharge, in front and rear, when the battery is properly planned with reference to it.

"(5.) Of the three factors of the efficiency of the stamp, the weight and fall determine the force of the blows at longer intervals. The blow must be heavy enough to crush the rock upon which it falls. If too heavy, it may waste power in packing the crushed rock; if too light, it may fail to crush, and so may pack. Finally, the speed should not be so great as to prevent proper clearance, or the stamp may strike a second blow upon the rock already crushed.

"(7.) The efficiency of a blow from a heavy stamp with short drop is less than that of an equal blow in foot pounds given by a lighter stamp with longer drop, because the longer drop gives greater final velocity to the stamp, and this tends to crush more and to pack less.

"(8.) The superior effectiveness of frequent blows lies in the fact that there is a limit to the amount of crushing which can be practically performed by a single impact upon a given quantity of rock distributed over a given surface. There is another practical advantage of high speed: if stamps are left, as it were, standing in the pulp between blows, the material settles around them and they 'suck' when the lift commences. A great deal of power is frequently wasted in this way, by not picking up the stamps before they become partially buried.

"(9.) But even if the efficiency of stamps were always exactly measured by the product of the three factors mentioned—that is, by the number of foot pounds delivered per minute (which is certainly not the case)—there would still be good reason for preferring rapid running. After the necessary stability and strength are secured, increased weight of machinery is an evil. If equal results can be achieved by substituting speed for weight, the change is advisable."

The quantity of water for the battery depends upon the quantity of sulphurets or black iron sands present in the quartz, and averages half a cubic foot per minute per stamp. It must, however, be sufficiently large to carry the crushed sands over the blankets without allowing them to permanently settle upon them—say, about 28 cubic ft. of water per cubic foot of rock crushed. For rock poor in gold the supply of water is increased, to increase the rate of working, so that the quantity amounts to 33 cubic ft. and upwards to one cubic foot of rock crushed. The foregoing proportions of 28 and 33 cubic ft. to 1 represent weights of the cubic foot of rock crushed, of about 125 and 108 lbs. respectively. In winter the battery water should be somewhat warmed so as to prevent congelation. To have less inclination of the blanket and more water, is preferable to the opposite condition.

Apron, Sluices, and Blankets.—On leaving the mortar, where a preliminary amalgamation on copper plates frequently takes place, the slimes pass over an apron of amalgamated copper plates, and by this means a considerable quantity of the metal is removed and prevented from passing down the blanket sluices which immediately succeed the apron. The width of the sluices for the copper plates is usually 15 in.—or 3 in. per stamp; and there ought to be two or three sets for each battery, so that during the cleaning of one set the stream can be turned on into the other set. The full width, therefore, of each copper apron per battery is 45 ins.

The bottom edge of the lower screen holes when new is 3 in. above the dies in the mortar. The water in the battery during the crushing has a wave motion along the screens. It is of the natural temperature while passing through the battery and over the blankets; but later, when passing through the rubbers and copper plates, it becomes somewhat warmed by the influx of the hot water from the amalgamators.

The slimes after leaving the sluices, *m m*, Fig. 5, which are covered with amalgamated copper plates, pass into sluices covered with blankets. There are three sets of these sluices for every 5-stamp battery, placed longitudinally. Each of the sets consists of two sluices made of ½-inch planed sugar pine boards, one 10½ ft. long, the other 5½ ft., with a drop of 3½ in. between them. They are from 16 to 17 in. wide, with sides of 2 in. in the clear, and have an inclination of 1 in. to 2 in. to the foot.

The upper sluice has two strong blankets, each 21 in. wide and 5¼ ft. long, the upper of which overlaps the lower about 6 in. The lower sluice has only one blanket.

The pulp flows over two of the three sets of blankets, the third being kept in reserve for use when washing either of the other blankets. The upper blankets, which catch the bulk of the gold, are washed every twenty minutes, the lower every two hours. The washing is performed in two tanks, which are used alternately. They are made of 1½-in. planed pine boards, having a horizontal section of 3 by 4 ft., and tapering towards the bottom. They are 2½ ft. deep, and are provided with inclined shelves for the blankets, and plug holes for the discharge of water after the settling of the blanket washings. The water used in these tanks is warmed in a heater by the waste steam.

The quantity of crushed sand passing over the blanket sluices of a 5-stamp battery ranges from 8 to 12 tons in twenty-four hours. The blanket washings, which consist of gold, sulphurets, iron, and quartz sand, vary considerably in weight with the percentage of the metallic contents of the rock.

The quantity of slime passing per foot of sluice width is about 0·7 cubic ft. per minute, containing about 3·1 pounds of solid matter per cubic foot of water. The blanket concentrations amount to about 2·5 per cent. of the original crushed ore, and have a specific gravity of perhaps 3·4.

The accumulations from the blankets collected in the wash-boxes are introduced in front of the amalgamators, from which they are gradually swept into the amalgamators by a current of clean water having a temperature of from 100° to 130° Fahr., or they are treated by pan amalgamation.

The materials for the blankets is specially manufactured for the mill trade, and is shorn on the lower side, having the nap on the upper. Its weight is eight-tenths of a pound per running yard.

Water required for Gold Milling.—The quantity of water required to work gold ores by the wet battery process is generally estimated as follows:—

For the boiler, 7½ gallons per horse-power per hour.
For each stamp, 72 gallons per hour.
For each pan, 120 gallons per hour.
For each settler, 60 gallons per hour.

If the water used in the battery, pans, and settlers be run into settling tanks, it can be re-used with a loss of about 25 per cent.

In making my calculations for the quantity of water required for a 30-stamp mill, I averaged the calculations as follows:—Half a cubic foot per stamp, per minute, equal to 30 cubic feet per hour; 30×24 hours makes 720 cubic feet per stamp, per day; 30 stamps×720 cubic feet makes 21,600 cubic feet water for the batteries.

Calculating the boiler for 60 horse-power, water required:—

One cubic foot per horse power per hour	= 60 × 24 =	1,440
Sixteen cubic feet per pan, per hour	= 16 × 24 =	384
Eight „ „ settler „	= 8 × 24 =	192
		2,016
For battery		21,600
Total in cubic feet of water		23,616

Taking it that the flow per miner's inch under ordinary six-inch pressure gives 2,160 cubic feet in twenty-four hours, this would represent 11 miner's inches; but it is safer to have a flow of 12 to 16 miner's inches before venturing on the erection of a mill of such capacity, and it is better to take the maximum figure than the minimum. A stream or water supply ought never to be gauged in the wet season, but the minimum flow of the dry season should be ascertained.

Amalgamation in the Battery.—In some mills amalgamation is performed in the battery, which is supplied with quicksilver for the purpose of taking up the gold when liberated by the crushing process. The quicksilver is usually introduced into the batteries in small quantities as may be necessary, having regard to the richness of the ore and the rapidity with which the amalgam is formed. The appearance of the latter as it issues from the battery is an indication of the quantity of quicksilver present. When the particles of amalgam are hard, dry, and dense, they indicate the absence of a sufficient quantity; while fluidity of the particles indicates the presence of a sufficient quantity, or possibly a surplus of quicksilver may be inferred. The ends of the batteries are lined with amalgamated copper plates, while another plate of the same character, about 10 or 12 in. wide, with a length the same as the inside of the battery, is so fixed in a frame that it may be introduced and secured behind the stamps in an inclined position. A similar plate, but narrower, is used on the discharge side of the battery. A portion of the amalgam, as it is formed in the battery and splashed against these plates, adheres to the amalgamated surfaces and is retained by them. The batteries and plates are cleaned at stated intervals, which vary in length in different mills.

A large portion of the total product is obtained by means of this contrivance, while the remainder is secured by other contrivances—namely, aprons or tables, covered with amalgamated copper plates—outside the battery. The inclination of the tables is usually 1 in. for every 1 ft. in length, being variously determined by different mill-men. It depends, of course, on the quantity of water used and other conditions. From these tables the material passes as before stated over blanket sluices, or rubbers, amalgamators, and in more modern mills, into the concentrators or shaking tables, or such other contrivances as the character of the ores may require, for further treatment.

When starting a new mill it is very difficult to obtain at once good results. The yield of the first crushing is very seldom satisfactory, and never comes up to expectation, as a fair per centage of the value of the ore is not obtained. A variable and sometimes large amount of gold must be absorbed by and remain upon the amalgamating plates of a new mill, and this item alone may considerably diminish the amount of gold obtained from the first run; but above all, the copper plates have to assume a proper working condition before they will do effectual work, and this takes several weeks, and sometimes months, as Californian experience has shown.

When starting a new mill it is always advisable, for the first twenty-four hours, to pass through the battery poor ores or waste, so as to fill up all the interstices and crevices in and around the dies and all other machinery through which the pulp has to pass, and the richer ores ought not to be crushed until the copper plates and mercury are in proper condition, so as to avoid a waste of precious metal.

The process of amalgamation in the battery is as follows: Quicksilver is introduced through the charging slit (N, Fig. 5) from time to time, generally once in two hours, and in quantities dependent upon the richness of the ore. Care is taken to distribute it as evenly as possible under the stamps. The quantity of the quicksilver employed is, on the average, twice as much as is afterwards recovered in the amalgam, the amount lost in milling being one-twentieth of a pound, and sometimes more, per ton of rock crushed.

The quicksilver, being finely divided by the stamps, acquires the opportunity to coat or amalgamate the particles of gold. This fine division of the mercury is proved by the fact that two-thirds of the quantity charged generally escapes in the battery slimes. Through the violent motion of the battery water produced by the fall of the stamps, the particles of gold amalgam or quicksilver are carried with the pulp upon the copper plates, to which they have an opportunity to adhere. The quicksilver which escapes through the screens is thinly fluid, and contains but a small percentage of gold, while the amalgam of the plates inside the battery is either a pasty or hard layer; that which is thrown out of the screens generally contains some gold, and catches on the outside plates, where it gathers more and more gold, and grows stiffer and harder in consistency.

The movement of the pulp in the battery consists in alternate rising above and falling below the inner plates, combined with wave motions longitudinally along the battery box. Outside, the pulp streams with uniform velocity over the inclined table—a movement which does not, like that within the battery, present moments of rest favourable to the deposition of the gold. The only place where this is at all the case on the outer plates is the upper end of the apron, upon which the discharge falls from a height of several inches, the plates being arranged in several steps or drops. It is a great mistake to have one flat uniform inclined apron of copper plates in front of the battery. The more steps there are, the better for a successful amalgamation. Over the smooth plates the stream of water carries the small particles of gold along, and not being able to reach the bottom and adhere to the plates they are lost. It is therefore important to catch as much gold as possible inside the battery.

It is obvious that most of the coarse gold will settle in the mortar and remain sticking to the inner plates; while the fine gold, on the other hand, is lifted from the bottom immediately after being set free from the ore, and is held in suspension, together with the particles of amalgam, by the battery water, until it is caught on one of the inside plates, or is discharged through the sieve.

While the substances are thus floating about, it is difficult on account of the thinness of the pulp to unite the particles of gold and quicksilver; and this explains the fact that the quicksilver which escapes through the screens contains little gold.

With a copper apron having an inclination of 10° and a width of ten to twelve inches per stamp, one-fourth of a cubic foot of pulp would be passing over it every minute. The discharge moves over them in a very thin layer, which moves much more swiftly on the surface than on the bottom. Such particles of the gold and amalgam as sink with the larger and heavier pieces of ore to the bottom slide or roll slowly along, and have opportunity to adhere; but smaller particles are swept along by the more rapid surface current of the watery sheet of pulp, without being able to reach the bottom. To this latter class belong the greater part of the gold, especially of that which escapes through the sieves, and almost all the particles of quicksilver and amalgam. Moreover, the absolute quantity of gold which can come in contact with the outer plates is less than that which may touch the inner plates, by the amount retained upon the latter. For all these reasons, the inner plates must furnish the greater part of the gold obtained in the mill.

It should be added, further, that coarse gold will either not at all, or very seldom, be lifted up to the discharge level inside of the mortar, unless it remains under the stamp and becomes pounded into such a shape as not to longer withstand by its gravity the swash of the battery water, or becomes amalgamated. It may thus become reduced into fine particles and be discharged. Coarse gold, at all events, stands a good chance to remain inside the battery or to be thrown against the inner copper plates and stick there; whereas fine gold will be thrown on to the outer plates, where conditions for amalgamation are not so favourable as inside the battery.

The gold which has not been liberated during the crushing, but remains in the rock, evidently becomes a loss, as it will flow over the outside aprons to find its way to the tailing pit.

Conditions of a Good Result from Amalgamation.—These may be stated as follows :—

(1.) Very fine stamping if the gold is very fine.

(2.) A coating of gold amalgam on the copper plates. No amalgam should be removed until a hard layer of it has been formed on the plates.

(3.) The use of a proper quantity of water; too much will result in a coarse crushing, a less complete mechanical exposure of the fine gold, less contact of the gold and quicksilver, and a premature sweeping off of both. The addition of too much quicksilver, though causing the formation of more amalgam, will only lead to its being swept away by the current of water.

(4.) Proper temperature of the battery water. The water ought neither to be too cold nor too hot: 90° to 110° Fahr. is the best, if by artificial means it can be raised to it.

(5.) The addition of quicksilver in proper quantity. This condition should be closely studied in each mill. There is always a considerable loss of quicksilver. The quicksilver when introduced in the battery is finely divided by the stamps, and thus affords an opportunity for the amalgamation of the fine particles of gold. By the violent motion of the water in the battery, produced by the fall of the stamps, particles of gold, amalgam, and quicksilver are carried with the pulp to the copper plates, to which they adhere.

(6.) Proper height of the charge in the mortar. This should not be allowed to rise higher than about three inches below the lower edge of the inner plates. If the quartz and pulp in the battery come nearer the plates, too much stuff, which is also too coarse, is thrown upon them, thus either preventing the accumulation of amalgam or displacing it after collection.

(7.) Regular feeding.

(8.) Care in keeping the plates clean.

(9.) Care in the mill against the introduction of grease or greasy substances, and against the use of exhaust steam for heating water required in any of the amalgamating processes. In lubricating the cam shaft, journals, cams, tappets, or any other portions round the batteries, care must be taken not to [illegible] any lubricant into the mortar.

(10.) Rejection of hydrated oxidised iron ores, silicates, magnesia, and alumina ores: they cause a frothing of the water and coat the gold with a slime which resists amalgamation.

(11.) Avoidance of mineral waters for battery amalgamation, especially if they contain sulphur in the shape of sulphuretted hydrogen, as a coating will be formed on the gold particles which prevents amalgamation.

(12.) Care that the amalgam on the copper plates is not allowed to get too hard, as it may fail to catch the gold. If it be noticed, therefore, that the amalgam is getting too hard, it will be well to sprinkle through chamois cloth some globules of quicksilver over the amalgam. If, however, it becomes too soft again, there is danger of "flouring" and losing it, with some of the gold.

(13.) A dilute solution of cyanide of potassium should always be kept at hand, and when yellow spots appear on the plate some of the solution should be poured over it. If this does not remove a spot, hold a lump of the cyanide over it, or rub it, which will have the desired effect.

(14.) If the ores contain soluble sulphates, arising from the decomposition of iron or copper pyrites, the addition of lime will prove beneficial while they are being passed through the battery.

(15.) When treating gold ores containing manganese, it is necessary to clean the plates of its adhering amalgam at least once a week and give them a fresh coating of quicksilver.

(16.) It will be found that in many cases amalgamation can be promoted by discharging from the battery on to concentrators direct, which will collect all the heavy mineral particles which interfere in the ordinary amalgamating process, and submitting the concentrates to separate treatment. The overflow or tailings which pass the concentrators, if carried over copper plates, will give up the gold easily now, in case any fine particles have escaped the concentrating process, as the ore has undergone a cleaning process in the separation of the sulphuretted ores, which naturally interfere with copper-plate amalgamation. If the concentrations are submitted to treatment in grinding pans, the tailings from the settler ought to be run over slime tables to collect any escaping mineral particles of value.

(17.) When treating heavy pyritic ores, it will be found advantageous to have an iron pipe with little holes, discharging on the outside copper plates a fine, gentle shower of water, so as to assist the carrying off of the heavy sulphurets, which would cover the plates over and prevent the free gold coming in contact with them.

In the next chapter will be found details of information on several matters connected with the working of the processes with which the present chapter has been concerned.

CHAPTER III.

TREATMENT OF GOLD-BEARING ORES: CRUSHING AND AMALGAMATION—continued.

CLEANING THE MILL—Cleansing the Quicksilver—Sodium Amalgam—Retorting the Amalgamated Metals—Workmen in the Mill and their Duties—Wear and Tear of a Mill and Cost of Milling—Results of the Mill Process—Loss in Milling—Mr. Paul on Waste of Gold in Amalgamation—Checking the Mill Returns—Mr. Skey on the Absorption of Sulphur by Gold—Conditions of Successful Working of a Gold Mine—Amalgamation of Concentrated Sulphur, &c.—Iron Battery Frames—Illustrations and Specifications of Batteries.

Cleaning the Mill.—The mill is cleaned every week. All the contrivances for amalgamating are usually cleaned every week, while the battery gold is removed once every few weeks, and on that day the mill is stopped for repairs. After removing the skimmings which float on the quicksilver baths of the amalgamators, the quicksilver is scooped out of the trough into a bucket and the amalgam taken up. The amalgam forms a layer at the bottom of the trough in the Attwood amalgamator, or at the bottom of the Hydrogen amalgamator. The upper trough contains 95 per cent. of amalgam in the apparatus. In order to free the amalgam from impurities, which consist principally of sulphides of iron, copper, and lead, precipitated with the sinking gold, it is worked by hand in a bath of quicksilver. The small lumps of amalgam are broken, and the impurities floating on the quicksilver removed by means of a cloth. After thorough cleaning, the quicksilver charged with the amalgam is pressed through a strong thick piece of canvas, and the remaining amalgam formed into balls of about 2½ in. diameter, weighing about thirty-five ounces each.

The quicksilver which has been removed from the amalgam in the troughs is allowed, unless a final clean-up is desired, to flow back again into the troughs without being filtered through canvas. The quicksilver only contains a small amount of amalgam. The quantity of quicksilver in the trough of the Attwood amalgamators is about 700 lbs.

The riffles are cleaned in a similar manner to the amalgamators. The amalgam is first removed by passing a small scoop slowly and closely to the bottom, and thus allowing the quicksilver to escape at the sides. The skimmings from the surface of the riffles are added to those of the amalgamators for treatment in the pan.

The amalgam on the copper plates is removed by means of a dull chisel, which operation must be carefully performed so as not to expose the copper. When the amalgam is removed, quicksilver is sprinkled on the plates and spread over it by means of a piece of rubber belting. The surface is finally washed with clean water.

The amalgam from the copper plates is freed from impurities by rubbing it in an iron mortar, with the addition thereto quicksilver, while a current of water kept flowing into the mortar carries off the impurities. In larger establishments, this operation is usually carried out in a small clean-up pan.

The scum, sand, sulphurets, &c., removed from the amalgam by washing, are treated in the amalgamating pan with the skimmings.

In cleaning the battery, the stamps are removed from the mortar by means of a block and tackle. The corner die, provided near the foot of the cylindrical body with a wedge-haped recess for the introduction of the crowbar, is taken out All the casings are removed in the same way. Dies and ings are raised and taken out of the battery. The battery hen cleaned and the sands washed by passing through a ve, having eighty-one to one hundred holes per square inch, to a wooden hopper, through which they are discharged into wooden sluice box; 6 in. wide and 6 ft. long. This latter is ovided with three or four riffle boards which catch the par- cles of gold. The sluice has an inclination of 1½ in. to the ot. The coarse sands from the sieve, after removing the iron by hand or by a magnet, are returned to the battery, while the finer sands which passed through the sieve are treated in the amalgamating pan; but if of a poor quality these also go back to the battery. The gold taken out of the riffles is freed from sand and iron by treatment in the pan and by the magnet, and is added to the amalgam.

While the mortar is empty, any worn-out dies, casings, and shoes are replaced by new ones. Before the old shoes and dies are returned to the foundry for recasting they should be examined for gold, which is frequently found in the crevices.

Cleaning the Quicksilver.—One of the essential conditions after a clean-up is made is to bring the quicksilver into an active, lively condition, and if possible to maintain it in this condition during the running of the mill. It takes very little to tarnish the quicksilver, and every mill-man has his own method of cleaning it. Some pour a little water on top of it, dissolve in it lumps of cyanide of potassium, and then stir up the quicksilver. Some use concentrated lye; others dilute sulphuric or nitric acid; but if quicksilver gets very foul and dirty, I prefer distilling it in the retort, which insures success, although a little troublesome and expensive in fuel.

Where electricity is obtainable for the purpose, the following method will work very effectively. Take a flat earthenware vessel, pour some sulphate of soda solution into it, and lay a long strip of lead across this vessel, in such a manner that it can be bent in the centre to plunge into the solution. Into this vessel place a semispherical non-glazed earthenware basin, large enough to fit the bottom vessel, and into this one pour the quicksilver with a little water. Put the positive wire in contact with the lead strip, which constitutes the anode, and the negative wire with the quicksilver. Evolution of hydrogen will take place, and the quicksilver will be seen to assume an active motion and agitation, with a brightening of the metal.

Sodium Amalgam.—This preparation—the invention of Mr. William Crookes, F.R.S.—has proved to be a very excellent ingredient in keeping the mercury active, although it has failed to accomplish the marvels which on its first introduction were anticipated from its use. Miners expected to extract the gold from iron pyrites, arseniurets, tellurides, by simply adding the sodium amalgam to the mercury in pan amalgamation, without roasting or any other chemical preparation.

Mr. Crookes gives the following directions for its preparation:—Amalgam A is a simple mixture of sodium and mercury in the proportion of 3 parts of sodium to 97 of mercury. The preparation of the mixture is as follows: Place a strong iron flask with a narrow neck nearly up to the mouth in sand on a sand bath kept at a temperature of about 300° F.; weigh out the mercury and sodium, put the former into the flask, and then add the sodium in pieces as large as a pea at a time, waiting for the action to cease before adding a fresh lump. The sodium had better be dropped in with a pair of tongs, and the hand should have a cloth over it. At each addition of the sodium a slight explosion is heard, and a bright flame will issue from the mouth of the flask. The action gets less violent as the mercury becomes richer in sodium. When the whole of the sodium has been added, pour the amalgam into a flat dish whilst still liquid, and when cold break it up and preserve in a stoppered jar. It will not require to be kept under naphtha. The amalgam forms a mass of long, needle-shaped, brilliantly metallic crystals, which interlace in every direction, but have very little cohesion. Amalgam B and C are specially prepared with an addition of zinc.

The inventor recommends the use of amalgam A in coarse powder. By sprinkling it over the wetted surface to be amalgamated, then rubbing it over with a little clean mercury, a firmly adherent and brilliant coating of mercury will be given to the metal. Not only can copper be amalgamated in this way, but the same result is produced on galvanised iron, tin plate, or lead, and less perfectly on iron and steel. In all cases it is advisable to preserve the effective surface and the amalgamating energy of the mercury on the metal plates by an occasional sprinkling of powdered amalgam A, applied from time to time as required.

I have found sodium amalgam very beneficial in pan amalgamation, as it has a tendency to keep the mercury bright and active. I never made comparative tests to ascertain whether by its use an increased yield of precious metal is obtained, but results published by eminent authorities indicate that a larger percentage has been obtained by adding the sodium amalgam to the mercury.

The Use of the Retort.—Amalgamation being complete, another process becomes requisite for the separation of the quicksilver and the amalgamated metals. The amalgam, which contains from 36 to 40 per cent. of gold bullion, is put in cast-iron cylinders or retorts. The cylinders are supported by brickwork on flanges, and are placed almost horizontally, having only a slight inclination toward the rear end, which gives the quicksilver, condensed on the front cover, an opportunity of flowing back to this end to be surrounded by the fire. The rim of the front plate or cover of the retort is provided with soft fine clay, and is firmly screwed to the rim of the retort by means of a clamp, so as to make it air tight.

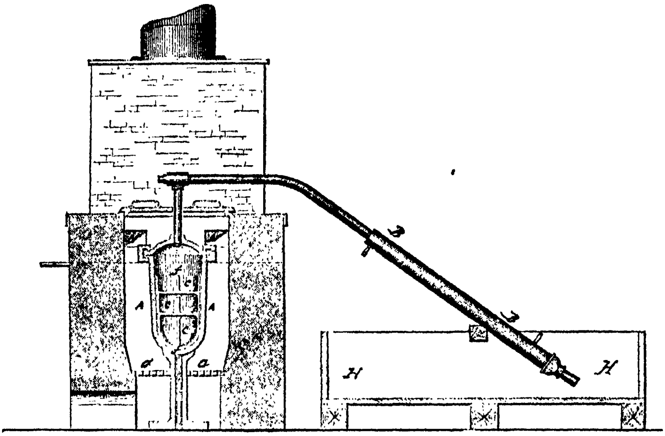

FIG. 45.—RETORT FOR LARGE MILLS.

Fig. 45 represents a style of retort used in large gold mills. The same contains three cups, having a tube in the centre for the escape of the quicksilver into the condenser B B. A A is the retort; *c c c* are the cups, *f f* are the central openings for the escape of the fumes; G G is the fire-grate; H H, wooden trough filled with water to collect the quicksilver; N N, flues leading into chimney.

A cylindrical horizontal retort of about 11 in. in diameter and 4 ft. 7 in. long will hold five trays, having a capacity to hold 1,350 ozs. of amalgam. The fire-grate necessary for a retort of this size measures 1 by 2 ft., with a chimney 4 by 8 in. in section. The quicksilver condenses in an iron pipe which is surrounded by cold water. The pipe is attached at the rear end near the top of the retort, and descends into a water basin. The heat, which is generated by wood in the grate under the front of the retort, passes through a return flue over the same into the chimney placed over the front of the retort. When the retort has been at a cherry-red heat for two hours, the retorting is considered complete. The time for this process, which varies with the quantity of amalgam in the retort, is from four to six hours, commencing with a cold furnace. After giving the retort time to become properly cooled, the front plate is removed. The gold bullion consists of a somewhat porous mass, and when cleaned has a bright yellow surface. It still contains a small amount of quicksilver, and a few sulphides.

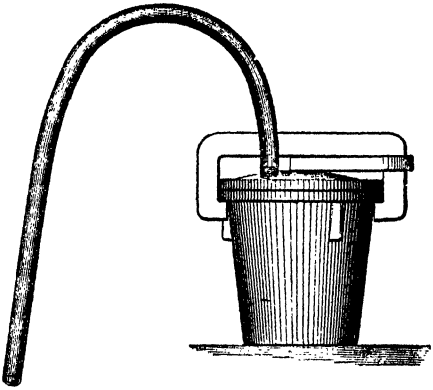

FIG. 46.—RETORT FOR SMALL MILLS.

Fig. 46 shows a small retort, such as generally used in small mills, by gold miners in general. The cover and retort are turned and ground to fit each other perfectly. The bale is of wrought iron, and takes hold under the flange on the retort; the wedge is then driven in between the cover and bale, making a secure and perfect joint.

The bullion is melted in black-lead crucibles, placed in air furnaces, usually 16 in. deep and the same in diameter, by means of charcoal. The fluxes chiefly used are carbonate of soda, borax, saltpetre, and sand. The loss in melting the bullion is 1 to 1½ per cent. The bars are from 700 to 940 fine, and contain, in addition to the usual silver, small quantities of copper, lead, and iron.*

Workmen in the Mill and their Duties.—One man is required at the rock-breaker during a 10-hour shift, and he generally has an assistant to remove the rock. One man will be required to feed each 10-stamp battery if no self-feeders are used. One man to attend to the washing of the blankets, and also to regulate the quantity of water passing over the blankets, to watch the proper discharge of the sieves, and to feed the blanket washings to the amalgamators, and to regulate the flow of water through them. There ought to be one blanket washer to every five stamps.

The attendance upon the amalgamating machinery devolves upon the amalgamator, who also prepares the screens, attends to the cleaning of worn-out shoes and dies, and various other minor duties in the retorting and melting department.

When steam is the motor, two engineers attend to the engine and boilers of a large mill in 12-hour shifts.

There is usually a night watchman.

There is also a general manager of the works.

Wear and Tear of a Mill and Cost of Milling.—In a well-managed mill accidents are of rare occurrence, but occasional breakages are unavoidable, considering the strong vibrations and jars to which all parts of the battery are subjected. Breakages are confined to the stems, the shoe shanks, and screens. The splitting of the tappet by wedging does not often occur.

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* See Chapter IX., on the Melting and Assaying of Gold.

The loss of quicksilver varies, but it is estimated to average 1 lb. for every 30 tons of rock crushed.

The wear of the blankets is over a yard a month.

A 5-stamp battery requires on an average 13 sets of screens a year. A set consists of 5 sheets of from 1 to 1¼ square foot.

To run a 30-stamp steam mill requires from 12 to 16 in. of water (miner's measurement), *i.e.* the discharge of an aperture of 12 to 16 square in. under a 6-inch pressure measured from the centre of the aperture. The water is usually supplied by ditches at so much per inch (miner's measurement).

It takes 5 to 6½ cords of fire-wood to furnish the steam necessary for a 30-stamp mill.

A good shoe lasts from twenty-one to forty-three days (on an average thirty-three days), and crushes 79 tons of rock. The wear is 1⅛ lbs. of iron per ton of rock.

The die lasts on an average seven weeks, crushing 100 tons. The wear is $\frac{6}{10}$ lb. of iron per ton of rock.

The stem generally breaks square across the fibres, near the upper face of the head, and lasts, without breaking, about sixty weeks, crushing 864 tons. When the irons are new and of fine quality the breakages are less frequent, occurring perhaps but once in a hundred and twenty weeks. Rewelding, including the necessary new iron, costs on an average something over £2. The stems as well as the cams last at least ten years. The battery linings last six months, and the tappets from two to three years.

The dies of rock-breakers, weighing 250 lbs. each, last about three months. The wear is one-half of their original weight.

The false pan bottoms last a year, and a set of four pan shoes three months.

The cost of a complete mill, including engine and boiler, is usually estimated at $1,000 (or over £200) per stamp. In a large mill having 20 stamps this includes the concentrating and the chlorination works.

The full cost of milling in a 30-stamp steam mill, the stamps weighing 850 lbs. each, with sixty-one drops of 10 ins., and having a crushing capacity of 72 tons per day, is $2·04 (or about 9s.), but this does not include the expense of concentrating the tailings.

In a smaller mill, having the same expense of engineers, the milling expenses are, of course, proportionately higher.

Mr. Hague * gives the cost of crushing in a small mill as follows, for a ton of ore:—

	s.	d.
Labour	5	0
Fuel	4	0
Castings and ordinary repairs	1	5
Quicksilver	0	3
Water, when purchased	0	10
Total	11	6

Mr. Reichenecker * gives the cost of crushing in California per ton, as follows:—

	s.	d.
Labour	3	2
Fuel and engineers	7	0
Castings and ordinary repairs	2	0
Quicksilver	0	6
Sundries	0	4
Total	13	0

Results of the Mill Process.—The gold realised by milling on copper plates from the battery is 60 per cent.; from the amalgamators 15 per cent.; and from the concentrates 25 per cent.

The fineness of crushing is as follows. The battery sands, crushed through a No. 6 slot screen, contain on an average—

(1.) Of slimes which remain suspended after a three minutes' rest in still water, 19 per cent. They contain when filtered the so-called "float gold," and consist principally of the earthy matter, mixed originally with the quartz.

(2.) Of slimes passing through a sieve of 6,400 holes per square inch (No. 1 excluded), 51 per cent.

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* Quoted by Professor Raymond in his Official Report on Mines and Mining.

(3.) Of sands passing through 1,600 holes per square inch (excluding 1 and 2), 23 per cent.

(4.) Of sands not passing through 1,600 holes per square inch, 7 per cent.

The blanket washings contain 78 per cent. passing through a sieve of 1,600 holes per square inch, and 36 per cent. passing through 6,400 holes per square inch.

The "concentrates" of the blankets contain, of course, a larger amount of heavy stuff. If the pulp discharges directly on to shaking tables or vanners, the amount of concentrates will depend on the character of the ore and the amount of sulphurets it contains.

In some mills as much as 70 per cent. is saved of the gold contained in the rock, especially when crushing ores which contain two or three ounces of gold per ton, and particularly if the gold is coarse and bright; but lower grade ores do not yield so much.

Loss of Gold in Milling.—The causes of loss of gold in milling (or the extraction of gold by copper-plate amalgamation) are manifold, and depend not only on the condition in which the gold exists in the rock, or on the minerals associated with it, but in many cases the earthy particles of the gangue exert a most detrimental influence in amalgamation and so cause heavy losses.

There are talcose and aluminous ores which during crushing will be pulverized so fine that 75 per cent. of a sample will pass through a 100-mesh screen. The slimes produced during such a trituration have an effect on gold like grease, and the fine particles of gold which are held in suspension in the battery waters surrounded with such fine particles of clay and talc are bound to be lost, as they are swept over the copper plates without adhering to them. These ores are very difficult to manipulate, as no concentrator has yet been constructed which will allow of the concentration of the mineral particles in such ores. Numerous lodes, also, are mineralised with sulphides, mostly so fine as to be invisible to the naked eye; and in such rocks the gold is generally present in a fine impalpable condition, it may be as free gold combined with sulphides. It is under conditions like these that difficulties in the economical treatment of gold ores arise.

One of the points to be closely watched and studied by the mill-man must be the conditions in which the gold exists in the rock. He may find one "pay shoot" richer in coarse gold than the other, and if he uses a 30-mesh screen for one he will do well to use a 40, 50, or 60-mesh for the other, as the screens must always be in proportion to the fineness of the gold particles. Liberation of the gold must be effected before it can become amalgamable with mercury, and as long as the silica surrounds the gold entirely it will prevent the precious metal from being attacked by the mercury.

In many cases gold said to be "rusty" is the presumable cause of loss, and although I have seen rusty gold obtained from placer mining, very little rusty gold has come under my observation in quartz mines. The rustiness of gold is caused by the coating of a metallic oxide whose nature is not exactly understood.

It is advantageous to ascertain in every mill how much gold goes into the tailings, and the quantity of float carried away by the water. Of course all mill-men try to crush the largest quantity of rock in the shortest possible time, so as to get a large output of gold. This practice makes the ore too coarse to allow all the gold and silver to amalgamate, as a portion may not be released from the gangue. It would be much better to get the output by a more careful sieving of the ore, not forcing the stamp to do the work of a Blake crusher, nor sending to the mortars any ore fine enough to pass the screens. This is a matter of some importance, for it has been found with all kinds of stamps which have screens that it takes just as long to get ore which has already passed the screens out of the mortar as it does to crush and force it out. Crushing too fine is also quite as bad a practice, as it produces "float," and is likely to put the precious metals in a condition in which they will not amalgamate. Even supposing that losses due to improper working do not exist, there are still a few causes of loss which, though not in every case amounting to much, were a source of considerable loss in the early days. It has been found that holes in the castings of the stamps, pans, &c., attract the amalgam, and that it is even carried into holes deep in the interior of the piece. This was a source of profit in the early days to those who recovered the precious metals from the worn-out castings. Another loss may be incurred in cleaning the plates by taking off the amalgam too thoroughly. It is a well-known fact that new plates do not act so readily as old ones. As mentioned before, gold and silver will be attracted better by amalgam than by mercury. Too slow a current of water will keep the surface of the plates covered with a film of sand, while a too rapid current will prevent the gold being caught by the plates.

If gold is attached to a piece of the gangue rock which is relatively large, the specific gravity may be so reduced as to prevent the particles coming in contact with the mercury.

If the blankets are kept too long without washing, so that the hairs become charged, the fine particles of gold are lost.

Even if these causes of loss are avoided there are still others; for instance, if the mercury be not kept clean, or be not made so by chemicals, the "quick," having an extremely thin film upon it, does not act upon the gold or silver. To some extent the same effect is produced when the rock is soapy, as is the case with the magnesian and aluminous rocks. If also there are too few amalgamating machines, or the sluices are too short, loss arises.

A very important source of loss is the "flouring" of the mercury caused by rapid motion, or due to a too free use of chemicals, or the presence of base metals.

Waste of Gold in Amalgamation.—There is an idea among the general public that gold is readily amalgamated, and that it is not necessary, therefore, to be so particular in its manipulation in the mill. How erroneous this impression is will be seen from the tests made by Mr. H. B. Paul, of San

Francisco, at some of the Californian gold mills. The following results * were arrived at:—

Test No. 1.—Average yield of ore in mill, $18·60; wastage after complete washing, including concentrating; silver, $3·14; gold, $10·04; total, $13·18.

Test No. 2.—Same mill tailings 350 feet from mill: silver, $3·93; gold, 5·02; total, $8·95, showing that a percentage secreted itself in the passage down stream.

Test No. 3.—Average yield of 150 tons, $3·50; assays of tailings carefully sampled: silver, $6·28; gold, $13·55; total, $18·83.

The unsatisfactory results were occasioned by the extreme fineness of the gold, and even the above does not show the full wastage. A large amount of gold produced is in such fine particles that it will float on water, and hence its name—"float gold."

In cleaning up the slum from a water-tank for supplying the battery, where the water was used over and over again in consequence of its scarcity, it was found that this slum—that is, the settlings of the very finest particles held in suspension by the battery water after the water had remained at rest in the tailing pit, whence it was pumped into this tank—yielded, by a rough amalgamation in a tub, $33 in silver, and $56 in gold.

Another test, made from fifty pounds of tailings taken one mile below the mill, showed that they contained 55 per cent. of what was the average working of the mill. A test, made on 1,500 lbs. of tailings, showed the loss in mill working to be 63 per cent.

The idea, therefore, that gold ores are readily amalgamated is, in truth, absurd.

It must also be considered that a very small percentage of lead, copper, arsenic, or antimony, such as is found in most gold ores, will quickly vitiate the mercury, rendering it quite inefficient in collecting even the gold that otherwise, from gravity, might be taken up. This renders it necessary that the mercury should always be kept in a clean and active condition.

* Quoted by Professor Raymond in his Report on Mines and Mining.

Checking the Mill Returns by Sampling and Assays. —It is a rather difficult matter to get an average assay of gold ores at the mill, for not only is the gold unevenly diffused throughout the rock, but considering that only half an ounce or one ounce is generally taken as the weight for an ore assay, a very small speck of gold more or less in the assay pulp will vitiate the result. At the best, therefore, only an approximate assay of the rock as delivered at the mill can be obtained when working.

My method of sampling the ore was usually as follows: as the rock-breaker discharged the ore, I would place a cup, holding about five pounds, every fifteen minutes under the discharge till full, and this would give 20 lbs. of sample in one hour, or 480 to 500 lbs. in twenty-four hours. This ore I would spread out on the floor of a cleanly swept room, and make a square of it 12 × 12 feet; then every foot I would draw a line across with strings tied to nails driven into the floor, giving twelve imaginary divisions 1 foot wide by 12 feet long; then transversely across these lines I would draw twelve other strings one foot apart, which would then divide the ore layer into 144 one-foot squares. Each square was then carefully sampled by taking as near as possible one pound of ore out of it on the point of a shovel, giving a 144-pound sample, which in a separate grinding machine was reduced to pieces about the size of a pea. This reduced rock was again spread on a clean floor and divided into 144 one-foot squares, and from every square 4 ounces taken. The resulting 576 ounces were ground into a fine pulp, again divided on a large table into 144 squares, and a quarter of an ounce taken from each; and from the resulting lot of 36 ounces—which should be the representative sample of the twenty-four hours' crushing—the half-ounce assay was taken and tested. The quantity of rock being known which had been passed through the mill, it was easy to calculate the quantity of gold therein contained.

Now attention had to be turned to the tailings which escape from the mill through the discharge launder. Every fifteen minutes I held an iron porcelain-glazed pot under the spout till full, taking care not to let it overflow; then I set it aside, and after fifteen minutes, before taking the next sample, I decanted—by preference with a syphon—the supernatant water, taking good care not to disturb the settled part, into a large wooden tub. This operation being repeated for twenty-four hours, the settlings in the pot were dried, sampled, and assayed. This was the tailing assay. Subtract the tailing assay from the o[illegible]y, and the balance represented the quantity and value of [illegible] gold which had been, or ought to have been, saved in the mill out of every ton. Very seldom did this prove to be correct, and if attention were now paid to the large tank containing the water poured off from the iron vessel, and this allowed to remain quiet for a day or two, then syphoned off, it would be found that the small amount of residue therein contained was rich in gold. This was the "float gold," the bugbear of mill-men and metallurgists.

The Absorption of Sulphur by Gold.—Mr. William Skey, analyst to the Geological Survey of New Zealand, has published an interesting paper on the absorption of sulphur by gold, and its effect in retarding amalgamation. While investigating the causes of the reported loss of gold during the process of extraction at the Thames gold-fields, he observed that much of this loss could scarcely be referred to any of those causes generally supposed operative for it. He therefore tested the actual condition of the natural surfaces of numerous specimens of Thames gold in respect to their behaviour with mercury, and examined further than had hitherto been done into its comportment with several of those substances likely to be associated with it in a natural way. He found—

(1.) That numerous samples of bright, clean-looking gold, of all degrees of fineness, refused to amalgamate on any part of their natural surfaces, though taken directly from the reef and untouched by hand.

(2.) That on such surfaces sulphur was always present.

(3.) That native gold, or gold in a pure state, readily absorbs sulphur from moist sulpheretted hydrogen or sulphide of ammonium, and absorbs it directly when administered in boiling water.

(4.) That surfaces so treated refused to amalgamate, though no apparent change could be observed in their aspect.

(5.) That gold so affected is rendered amalgamable by roasting in an open fire, unless copper is present to the extent of seven per cent., or perhaps less, while the same effect is produced by the contact of cyanide of potassium, chromic and nitric a[illegible]d chloride of lime acidified.

(6.) That this absorption is altogether of a chemical nature.

(7.) That sulphates of iron, in presence of air and water, decompose various metallic sulphides common to auriferous reefs in such a manner as to liberate sulphuretted hydrogen.

The action of sulphuretted hydrogen upon gold, in rendering it non-amalgamable when placed in contact with mercury, was demonstrated with striking effect by actual experiments.

From these results Mr. Skey was led to suppose that a large area of the natural surfaces of native gold is covered with a thin film of an auriferous sulphide, and that the greater part of the gold which escapes amalgamation at the battery is represented by that portion of this sulphurised gold which has remained unabraded during the processes of milling or extraction from the reef; the state of the gold, rather than that of the mercury, being therefore the greatest impediment to thorough amalgamation.

In addition to these results, he communicated others relative to the effect of solutions of sulphuretted hydrogen and sulphide of ammonium upon platinum. In rendering it non-amalgamable, he believed a sulphide of the metal had formed in each case, since chromic acid rendered it again amalgamable. He further stated that this metal is so affected by ammonia or the fixed alkalies that it will not amalgamate except in presence of a mineral acid, from which he suspected that platina is capable of superficial oxidizement when in contact with alkaline substances, even at common temperatures. He found that his samples of gold were not affected by the alkalies in this manner, except in the case of one from Victoria, a singularity from which was argued the presence of palladium in this particular sample.

Conditions of Successful Working.—The successful working of a gold mine depends at the present day on a perfect and rapid system of "concentration." Mines which carry only £1 of free gold can be worked at a good profit if the local conditions are such as to permit of easy mining and subsequent reduction in the mill. As the attention of inventors is now specially directed to improving the concentrators, so that the lowest grade ores may by washing away the waste be converted into high grade ores, and as cheaper methods of reducing the concentrates are being developed, the field for mining operations will be almost daily enlarged.

A description of the concentrators now generally in use in different countries is reserved for a later chapter.

Amalgamation of Concentrated Sulphurets after roasting.—One would have expected to find in existence works where the concentrated auriferous sulphurets and arseniurets could be submitted, after an oxidising roasting, to pan amalgamation, as every experienced roaster will have observed that pyrites, after a perfect dead roast, will show pure, clean gold on horn spooning, or panning down a sample. Such gold is amalgamable gold, and will yield 95 per cent. by pan amalgamation. "Amalgamable" may not be a correct metallurgical expression, but it will, I think, convey to the reader the idea I desire to express—viz., gold which can be absorbed by quicksilver. Although, however, the desulphurizing and amalgamation of gold sulphurets has for many years been extensively experimented upon in California, no plan has so far been brought into general use whereby refractory ores are satisfactorily treated, or low grade sulphurets worked with profit by pan amalgamation, the only tolerably successful method for reducing sulphurets being by the chlorine process, which is much too expensive to answer for low grade ores.

In the early days of Californian gold quartz mining, amalgamators used to grind in pans without roasting, contending that gold, being found in a metallic state, only requires grinding to a certain degree of fineness to admit of amalgamation; but this theory was not verified in practice. If the gold is in a free state in the sulphurets (a point by no means cleared up as yet) however fine the gold may be, it is coated with sulphur, iron, or some other metal, and no amount of friction will brighten it sufficiently to become amalgamable.

The best result which can be obtained by long and continued grinding of raw (unroasted) concentrated sulphurets is 40 per cent., and no doubt the greater part of the gold thus obtained is contained in the concentrates in a free state. To treat sulphurets properly, therefore, they must be completely desulphurized, and all metals in them (except the precious metals) thoroughly oxidized before the silver and gold will amalgamate in pans. If a complete oxidation of the iron and copper is not effected, on being charged into the pan and submitted to the grinding operation, they will pulverise into a fine powder, and when the quicksilver is added, it will partly flower and float on top of the water, while the other globules of quicksilver, distributed through the pulp, will become coated with a black scum, thereby losing all its affinity for the precious metals. When the ore is thoroughly oxidised, no such "sickening" of the mercury takes place, and the gold is taken up. Ores which have been roasted concentrate very easily, as everything in them, save gold and silver, loses specific gravity.

The difficulty in treating large bodies of low grade iron pyrites by amalgamation is the prohibitive cost of the roasting operation, but with the introduction of mechanical roasters, which would desulphurise the pyrites completely at a reasonable ate, I see no reason why amalgamation should not hold the eld against chlorination.

Iron Battery Frames.—These are manufactured especially in England for exportation to countries where heavy timbers are scarce. It is better to ship to such localities an iron battery frame, which can be easily erected on the spot.

The following illustration (Fig. 47) shows an iron battery of 20 stamps.

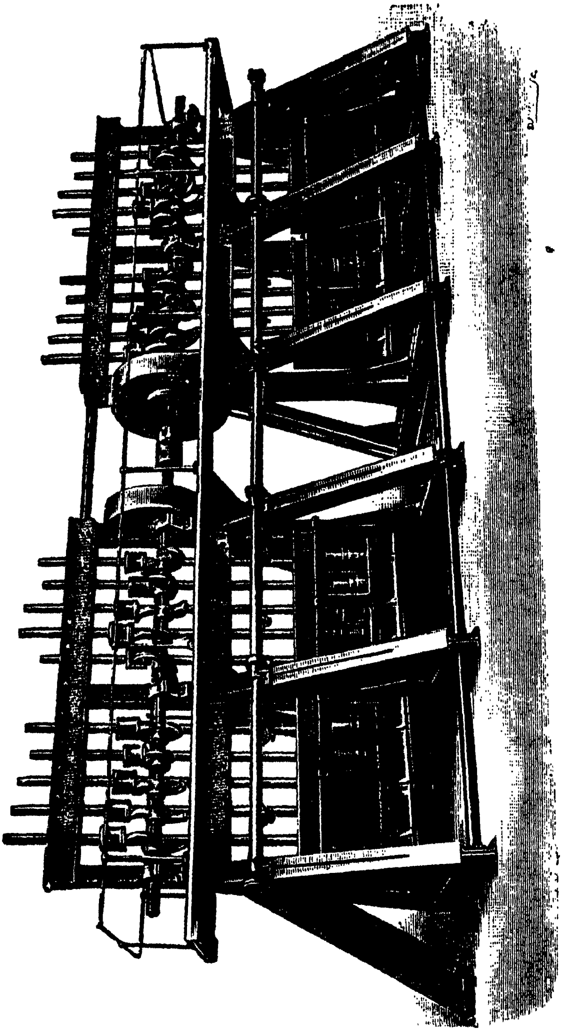

Fig. 47.—Twenty-Stamp Iron Frame Battery.

Battery with Ten Stamps.—Fig. 48 represents the battery when erected in its position in a wet crushing mill, The countershaft, and also its driving pulley with belt, are seen behind the main posts. Another pulley is shown by which the stamps may be stopped or put in motion without interfering with the driving power. The pulley on the cam shaft is of wood on cast-iron flanges. This mode of construction is necessary, inasmuch as a pulley made of iron, when subjected to the rapid succession of jars caused by the falling of the stamps, would soon break. The guides for the stamp stems are made of oak, maple, or some other hard wood. They are in two parts, and are bored at proper distances for the

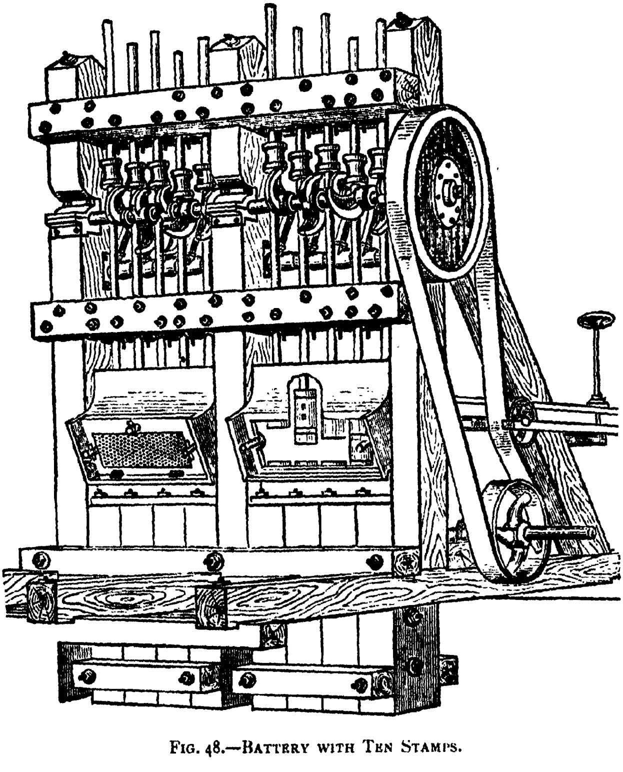

FIG. 48.—BATTERY WITH TEN STAMPS.

stamp stems. The guides, being set with keys between, can be closed together as they wear.

General Specification for a Ten-stamp Battery.—Two high cast-iron mortars of latest improved pattern, single (or double) discharge, planed on bottom, drilled by template, with the seats for the screen frames planed.

Two screens of hard wood fitted to the mortars.

Four wrought-iron keys for the screens.

Two screens of Russia iron or wire cloth.

Two sheets of rubber for mortar foundation, ¼ inch thick.

Ten stamp dies.

Ten stamp shoes.

Ten stamp heads or bosses, bored for stems.

Ten stamp stems of refined iron, both ends being tapered and fitted to the heads.

Ten stamp tappets, fitted with wrought-iron gib with two steel keys in each.

One cam shaft of hammered wrought iron, turned full length, key-seated and marked where cams are to be fitted.

Three cam shaft boxes, babbitted, bored, and planed on bottom and back.

Two cam shaft collars of wrought iron, with steel set-screws.

One cam shaft pulley, built up complete on double cast-iron sleeve flanges; to be thoroughly built and turned true, with the flanges keyed to the cam shaft.

One set upper hard wood guides for stems, bored for stems and bolts.

One set lower hard wood guides for stems, bored for stems and bolts.

Two jack shafts of wrought iron.

Four side boxes or brackets for jack shafts.

Ten cast-iron sockets for levers.

Ten hard wood levers for stamp holders.

Machinery for a 10-Stamp Gold Mill.

One No. 2 Blake crusher, 10 in. by 7 in.

One grizzly or ore screen, 3 ft. by 10 ft.

Two automatic ore feeders, if such are used.

Ten stamps of 850 lbs. each in one battery, including all ironwork, wooden pulley, and hard wood guides for stamp stems.

One set of water-pipes for battery.

Two copper table plates, 54 in. by 8 ft. by $\frac{1}{8}$ in. thick, pure copper and free from flaws.

Four copper lining plates for mortars, $\frac{3}{16}$ in. thick, of same quality.

One amalgam retort and condenser.

One countershaft for stamps, with bearings and pulleys.

One countershaft for crusher, with bearings and pulleys.

All necessary belting and lace leather.

One engine, 9 in. by 14 in.; 20 h.-p.

One boiler, 40 in. by 10 ft. complete.

One feed-pump, with belt.

One heater and all pipe connections.

Stamp Batteries.

List of sizes, complete in detail, as per above specification. Weight of iron work, including wood pulley and guide.

5	stamps,	450	lbs. each	9,500	lbs.
10	,,	450	,,	18,000	,,
5	,,	550	,,	10,000	,,
10	,,	550	,,	19,500	,,
5	,,	650	,,	11,000	,,
10	,,	650	,,	21,500	,,
5	,,	750	,,	12,500	,,
10	,,	750	,,	24,500	,,
5	,,	850	,,	15,000	,,
10	,,	850	,,	29,500	,,
5	,,	950	,,	17,000	,,
10	,,	950	,,	33,000	,,

Battery Frames.

One complete frame for 5 stamps for 550 and 650 stamps, weight 9,600 lbs.

One complete frame for 5 stamps for 750 and 850 stamps, weight 12,000 lbs.

One complete frame for 10 stamps for 550 and 650 stamps, weight 17,000 lbs.

One complete frame for 10 stamps for 750 and 850 stamps, weight 20,000 lbs.

Power required for a 10-*Stamp Wet Crushing Gold Mill.*

One No. 2 Blake rock-breaker ...	6	horse-power.
Two ore feeders	0	,,
Ten stamps, 750 lbs.	12	,,
Four Frue vanner concentrators ...	2	,,
One grinding pan, 3 ft. diameter ...	3	,,
One settler	3	,,
Friction	4	,,
Total	30	,,

Power required for a 20-*Stamp Wet Crushing Gold Mill.*

One No. 2 Blake rock-breaker ...	6	horse-power.
Four ore feeders	0	,,
Twenty stamps, 750 lbs.	23	,,
Eight Frue vanner concentrators ...	4	,,
One grinding pan, 3 ft. diameter ...	3	,,
One settler	3	,,
Friction	7	,,
Total	46	,,

CHAPTER IV.

MILLS IN OPERATION—A NEW MILLING MACHINE.

ZEILE MINE AND MILL—Providence Mill, near Nevada City—Father de Smet Mill, at the Black Hills, Dakota—The Huntington Milling Machine.

A DIGRESSION may be allowed in the present chapter, to enable me to give some particulars of certain milling establishments which may be useful to those who seek information as to the laying-out and arrangement of mills for the treatment of gold-bearing ores; and to these particulars I will append an account of one of the numerous machines which of late years have been designed with the view of superseding the stamping-mill which I have described in previous chapters.*

Zeile Mine, in Amadar County, California.—As an example of what can be done with proper management, I give a brief account of this undertaking. The mine has been opened properly, new hoisting works have been erected, and a new 40-stamp mill put up, which crushes 100 tons of ore per day. The ore body, on the 200, 400, and 600 ft. levels, in places is very large, reaching a maximum of 50 ft. between the walls, and will average 40 ft. The rock is low-grade, and is assorted in the mine, the best being sent to the mill and the balance thrown aside in the workings under ground. It is by a careful system of working that the mine is made to pay. The rock yields only $2, or 8s., per ton in free gold, and from $5 to $6 per ton in sulphurets, which brings the yield up to a fair average.

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* It may interest the reader to learn that in my work on the "Metallurgy of Silver" will be found an illustration, drawn to scale, of the fifty-stamp gold and silver mill erected for the Montana Company, Limited, at Marysville. The illustration shows the mill in elevation, plan, and section.

Attached to the mine are chlorination works, which are kept running day and night, working from four to five tons of sulphurets every twenty-four hours, yielding large returns.

Providence Mill, near Nevada City, California.—This may be cited as an example of a model gold mill. The new mill, which was erected in 1882, works admirably. It is located, with the hoisting works, on the steep hillside above Deer Creek, a situation which gives every facility of fall, so that from the time the ore is brought to the surface and goes through the various stages of crushing, amalgamating, and chlorination, it is constantly descending from point to point with scarcely any handling, except by machinery.

The mill, which contains 40 stamps, is connected with the hoisting works by a tramway, over which the loaded cars are taken. These cars drop their loads of ore on inclined gratings, called "grizzlies," through which all the quartz and fine stuff, not exceeding two inches in size, falls into the self-feeders on the floor below, one of which is placed in connection with each battery of five stamps, and feeds the battery automatically. Such quartz as will not go through the grizzlies is thrown into rock-breakers, where it is crushed. One man is required during the day to attend the rock-breakers, two in number, and this is all the manual labour required to feed 40 stamps, as the self-feeders will do that work as well as if it was done by hand.

When the crushed quartz leaves the batteries, it passes over the usual aprons and inclined sluices, and from thence on to the Frue concentrators, sixteen of which are used in the mill to separate the sulphurets from the sands. All the work of attending the sluices and concentrators below the batteries is done by one man, so that in this large mill of 40 stamps all the ordinary manual labour required is performed by two men. The sulphurets are taken from the concentrators as often as required, and removed to a drying room adjoining, and from thence run into the chlorination works, where they are worked in the usual manner, and both the gold and the silver secured.

The power used to drive the mill is water, 100 in., brought from the north side of the creek, through a 15-in. pipe, and under 390 ft. pressure. This is discharged through a 1¾-in. nozzle upon a Collins hurdy wheel, 6 ft. in diameter, which drives the whole machinery with the greatest ease, and apparently with plenty of power to spare. There is an automatic regulator which lets on more or less water, as may be required, when less or additional power is required by the rock-breakers. The mill throughout is a marvel of simplicity, convenience, and neatness, and may be said to run itself. The driving power is inclosed in a little box placed outside in an extension of the building.

This mill, with its cheap power and the reduction of hand labour to a minimum, seems the perfection of quartz milling, and its economy is seen in the fact that the cost of milling is reduced to 55 cents (2s. 3d.) per ton, including wear and tear and keeping the works in repair. This is the cheapest milling I heard of in California. The Sierra Buttes and Plumas Eureka Companies mill their ore at the same cost, but they own their own water, while the Providence Company have to buy theirs. The latter, therefore, can claim to be doing their work at a smaller cost.

Father de Smet Mill, Black Hills, Dakota.—As this mill, which was designed by Mr. A. J. Bowie, involves some new and important features which have given excellent results in practical working, the particulars I am enabled to give deserve special attention from those who have large bodies of free milling ore to treat.

The mines of the Black Hills, in Dakota, being of what is termed "low grade," their successful working necessitated cheap milling, which could only be accomplished with large and economically running mills. To insure the greatest possible economy, the Father de Smet mill, which was designed and constructed under the immediate supervision of Mr. Bowie, at a cost of about £20,000, was built on a plan different, in some respects, from any previously erected. The arrangement of the building in particular was novel, and the special objects kept in view in the arrangement were four, namely: (1) the constant supervision of the batteries, tables, and sluices, and consequent safeguard against robbery; (2) free access to all parts of the machinery, and room for handling the same; (3) very large ore bin capacity, to insure steady milling; and (4) the least manipulation of the material in all stages of reduction.

Among the results attained by this construction of the mill, besides those hereafter mentioned, the following may be briefly noted. The building has been thoroughly tested, and at times there have been one and a half million pounds of ore in the bins without any resulting damage or straining of the structure. The rock breakers, which are 53 feet above the battery floor, have been run to their full capacity, with the whole mill in operation, without causing much, if any, vibration in the building, or even on the car track at the top of the main division. The carpentry of the building, in fact, is such as to insure perfect bracing of the superstructure with solidity of foundation. Eighteen persons only are required per day to run the mill to full capacity.

The Building.—The entire structure covers an area 140 feet long and 60 feet wide. The elevation of the main building, from the mud-sills to the apex of the roof, is 75 feet. Its greatest length is east and west. It was constructed with three distinct compartments, which for present purposes may be designated as the east division, main building, and west division. The whole structure is of timber and lumber.

The east division is 52 feet long and 60 feet wide, and contains the engine, boiler, machine and lathe rooms. The machine shop is well supplied with all the necessary implements for mill work. The main or centre division, which is 68 ft. by 60 ft., contains the batteries, rock breakers, and ore bins. The west division, 20 ft. by 60 ft., is used for the clean-up room and repair shop. It contains a small cast-iron clean-up pan, tubs for panning out amalgam, tailings, &c. There is an ample supply of water for all purposes.

Engine, Boilers, Shafting.—The various machinery of the mill came from San Francisco, California. The engine is a horizontal one, fitted with Meyers's cut-off, known as size No. 5. The cylinder is 20 in. diameter with 42 in. stroke; the shaft is 9 in. in diameter, and the whole weight of the engine is 25,000 pounds. The balance-wheel is 18 ft. in diameter, cast in eight segments, and weighs 14,360 pounds. The engine is of sufficient capacity to drive the present eighty stamps and twenty additional, should the latter be required. There are two boilers 54 in. in diameter, and 16 ft. long. They are tubular boilers, provided with a 12-ft. steam drum and all the necessary accessories. To supply the boilers, and for other purposes, there is a No. 5 Knowles steam pump.

The main shaft and driving pulleys are situated in the centre of the east end of the main building. The pulleys (96 in. in diameter), by means of six-ply rubber belts, drive 54 in. pulleys on the line shafting; and eight other pulleys, also 54 in. in diameter, on this line of shafting are belted to 72-in. pulleys on the cam shafts. The cam shaft belts are provided with tightener pulleys 15 in. by 16 in. in size. There are two lines of line shafting, one on each side of the main building, placed behind the batteries, and directly on the battery sills, the head ends being supported on pillow blocks. The machinery and line shafting are so placed as to be readily accessible.

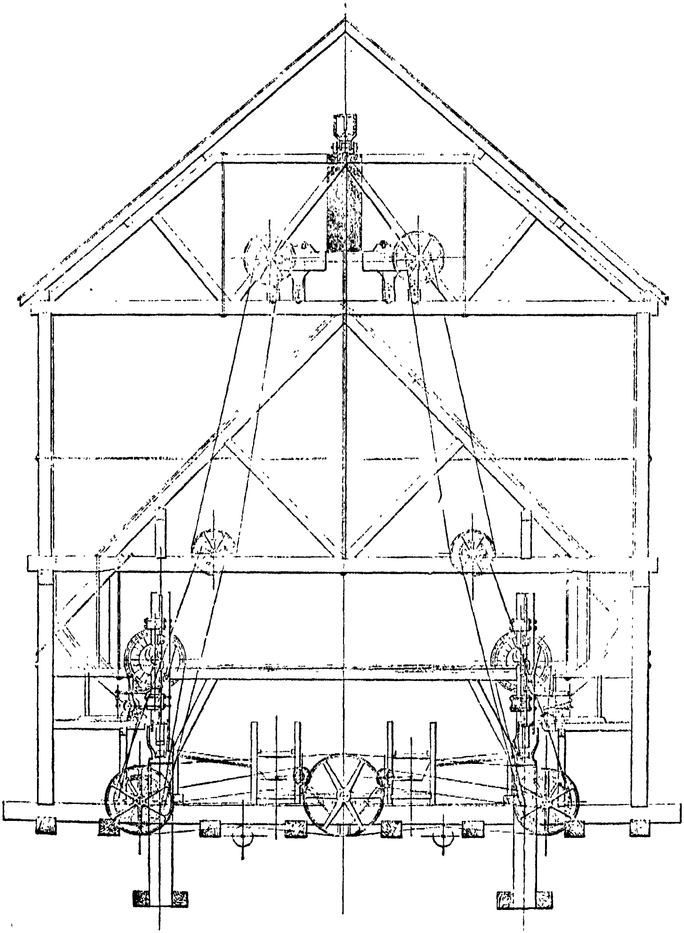
The Batteries.—The batteries, sixteen in number, containing five stamps each, are arranged in two lines, eight batteries on each side of the mill. They discharge to the centre of the main building. They are of the usual style, built independently of the building, and braced, one line against the other, with 12 in. × 18 in. timber. Each stamp weighs 758 pounds. The tappets are set to drop from 7 to 9½ in., 85 drops per minute being the calculated speed. The cam shafts are 5 in. in diameter, 14 ft. 6 in. in length, each weighing 860 pounds.

Mortars, Shoes, and Dies.—The mortars used here are of the gold mill pattern, with lining for sides and ends. The frame and lining weigh about 4,900 pounds each. They are set on the blocks and bolted to them, the tops of the blocks being previously covered with blankets. In the bottom of the mortar is a longitudinal groove, through the central axis, made for the purpose of holding the die to its place, each die being cast with a corresponding lug. As the dies wear down, it was intended to introduce a false bottom, or heavy casting 3 in. thick, filling the bottom of the mortar, and provided with a lug for the groove in the mortar, and a groove in the top for the lug on the die. The object in view was to economise iron by wearing the dies as thin as possible without lowering too much the level of their surface. The screens used are No. 6, punched, size 14 × 52 in.

Experience has since shown that, with the frequent introduction of dies, the sides of the grooves chip, and the dies then require wedging. It has been further demonstrated at the mill that, when old dies have been worked down on the false bottoms, the batteries by no means crush as much rock as with new shoes and dies, though the line of discharge is kept relatively the same. A very important question in milling low grade gold quartz is here presented, where economy requires that the largest possible quantity should be treated at the smallest possible cost in the shortest time, with due reference to the percentage of gold.

The conclusions so far arrived at are: (1) that the square bottom is the best form of die, and (2) that there is no economy in using dies after their surfaces have been irregularly worn to any great extent.

In milling gold quartz, almost the principal expense is the consumption of shoes and dies, and any method by which a saving can be effected here is most desirable. If the above conclusions are correct, it is economy to cast shoes and dies with the smallest depth practicable. This necessitates at least monthly replacing, but it insures the largest amount of work per stamp with the smallest consumption of iron. In addition, it is estimated that the quantity crushed with new shoes and dies is so much in excess of that crushed with those which are much worn as to well repay frequent renewal.



NOTE.—*This section shows the mill with the Eureka rubbers as originally c*

FIG. 49.—FATHER DE SMET MILL. Cross Section through Main Building.
Scale 1$\frac{1}{8}$ in. = 1 ft.

Feeders, Tables, and Traps.—Immediately over the shafting on each side of the mill is erected the feeder floor. This floor is spacious and well lighted. There are sixteen of Hendy's patent self-feeders, eight on each side. They stand immediately at the back of the batteries, the lip of the feeder being on a line with the mortar feed, and the mouth of the hopper of the feeder directly under the ore bin discharge. The batteries, sixteen in number, with five stamps each, are arranged in two lines, eight batteries on each side of the mill.

In front of each battery there is an inclined table covered with amalgamated soft copper plates. The tables are so located that they are all visible from any one point on the main floor of the mill, and are thus constantly under the supervision of the amalgamators. At the lower end of each table there is an Eureka rubber, which receives the tailings, and in turn discharges them into a quicksilver trap. From the trap, the tailings pass over a small amalgamated copper plate placed below the floor, and thence run into the sluice previously described, situated in the centre of the mill. The traps are narrow wooden boxes with a centre partition which extends to within a few inches of the bottom. Above the main sluice, on a level with the floor, there is a track on which runs a small car, used to transport material as may be required.

Rock Breakers.—There were two rock breakers when the mill started, the ore descending on grizzlies. The monthly capacity of the mill was originally from 4,400 to 4,500 tons of quartz. Two additional rock breakers have since been added, the capacity of the mill being thus increased to 6,200 tons of quartz, and to a much greater amount—7,500 tons—when the auriferous slate also is milled, which is now being done.

It has been conclusively proved in the Black Hills that in milling low grade quartz there is great economy in large rock breaker capacity. The breakers should be set to crush fine.

The Huntington Milling Machine.—It has been the aim of inventors for years past to produce a machine which in efficiency should equal, or even excel, the stamping mill; and although numerous attempts have been made, so far none have proved efficient enough to supersede it, at least not for wet crushing as usually required in gold mills. For dry crushing, especially where the ore has to be roasted, it has been found advantageous to use rolls, and those made by Krom meet the requirements best. Amongst the machines which have been designed to supersede the stamping mill, and which have advanced beyond the experimental stage, the Huntington mill deserves to be mentioned. The special merits of this machine seem to lie in economy in expense of plant and cost of working; in the economical transportation of machinery, and its erection at the mine; and in economy of management and power. The machines are made from 3½ ft. to 5 ft. diameter, weighing, according to size, from 2¼ to 5 tons; but for want of authentic data, I am not able to furnish their crushing capacity as compared with the stamping mill.

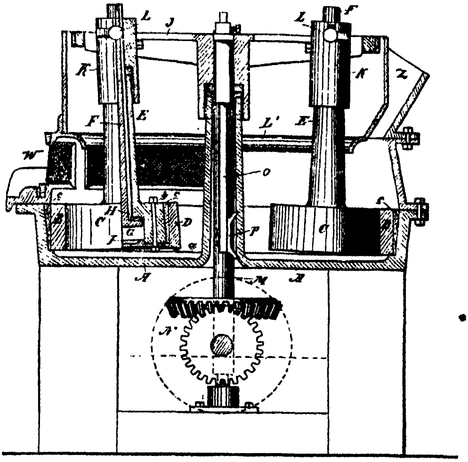


Fig. 50.—Huntington Milling Machine. Section.

The Huntington mill consists of a circular iron pan, in the centre of which through a hollow cone passes a vertical axis, which imparts a rotary movement to a circular disc forming (so to say) a cover to the pan; and from the periphery of this disc are suspended four vertical shafts, with rollers turning loosely upon the lower ends of the shafts, which are so suspended that the rollers may swing inward and outward, and be caused to travel around in contact with the sides of the pan or dies by centrifugal action. A set of scrapers are also attached to the disc, which throw the ore constantly in the path of the rollers.

The details of the construction of the machine are shown in the figures, which also give a perspective view. Fig. 50 is a sectional view, Fig. 51 a plan, and Fig. 52 the scraper.

The plan shows only two of the rollers, with the mode of suspending them from the periphery of the disc, but in practice four rollers are in the machine, as can be seen in Fig. 53.

A is a circular pan, having a concentric, B, of hard steel fixed around its interior in the portion which forms the bottom pan, which has a portion of its front reserved for a discharge opening, covered with a screen as shown at W. The feed slit is at Z, into which an inch pipe discharges the water necessary for carrying off the pulp.

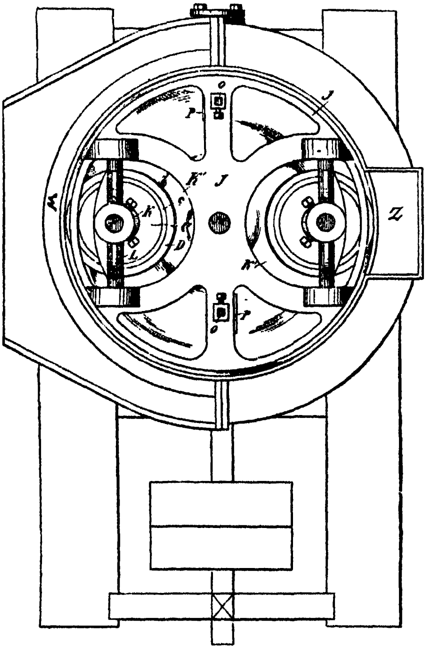


FIG. 51.—HUNTINGTON MILLING MACHINE. Plan.

The rollers, C, which do the crushing, have exterior circular shoes, D, secured to them. These shoes roll against the interior of the die, B, so that the material is crushed between them, and the wear comes upon the shoes and dies, which can be easily replaced when worn out.

These dies are rolled steel, and are formed with a flange, *a*, around their interior near the bottom, and hook bolts, *b*, pass through the rollers, C, with the hook beneath the flange, so as to support the dies and hold them in place. Wooden wedges *c* are driven between the dies and the rollers, and as they swell when wet, they will hold the two firmly together.

The flanges, *a*, are held by the hook-bolts against the bottom of the rollers, and they prevent the dies from being forced up by pieces of rock which may be jammed below them. The dies around the interior of the pan are secured by wooden wedges in a similar manner.

A sleeve, E, extends upward from each of the rollers, C, and has a hole made through it to the shaft, F, by which the crushing roller is suspended, to pass down through the sleeve and through the roller. The lower end of this shaft has an enlargement or head, G, formed upon it, and a corresponding chamber is made in the bottom of the crushing roller to receive this head. Between the head and the interior of the chamber, washers, H, are placed, which serve to relieve the parts of friction.

A cap or plate, I, is fitted with rubber or other packing so as to screw upon the bottom of the crushing roller, thus making the chamber within which the head of the suspending shaft lies perfectly tight; and this chamber is then filled with oil or lubricant so that the rollers may turn easily upon the shaft, and at the same time be kept entirely free of the grit and dirt. J is the driving disc, from which the rollers are suspended.

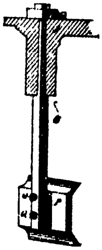


FIG. 52.
SCRAPER.

K is a sleeve, having trunnions, L, projecting to each side from it, and these trunnions turn in boxes supported by the lisc, so that the trunnions stand at right angles with a radial ine from the centre of the disc.

The upper end of the shaft is keyed in this sleeve, and the ower end of the sleeve has an enlarged opening which fits over upper end of the sleeve, E, as shown. This prevents any irt from falling in at the top of the lower sleeve and working own around the shaft, thus keeping the whole of that portion ntirely clean and well lubricated.

The disc, J, is peculiarly formed, with large curved openings , so that when the trunnions are lifted out from their journal boxes the shafts, with their sleeves and the attached rollers, may be lifted directly out through these openings without in any way disturbing any other part of the apparatus.

By this construction the rollers are allowed to swing inward and outward about the suspending trunnions, and as the disc is driven around by the central driving shaft M, to which it is keyed, and the gearing, N, below the rollers, C, will be caused to rotate about the shafts, F, rolling against the interior dies which are fixed within the pan, thus crushing all the material which may come between them until it has reached a suitable fineness.

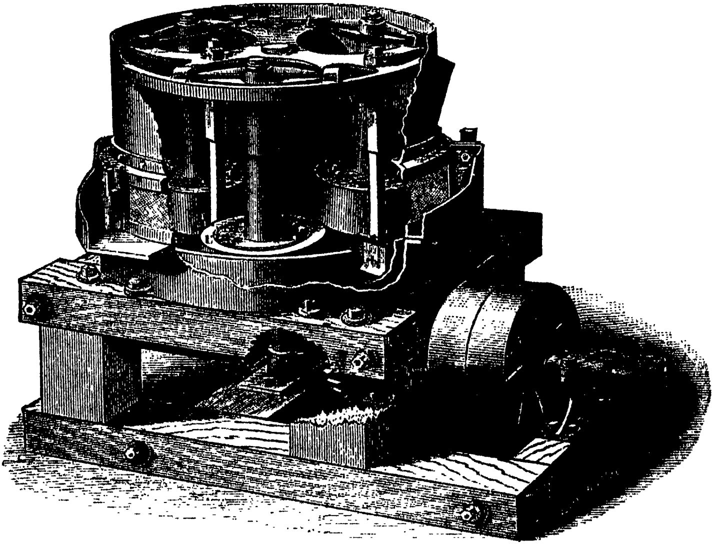


FIG. 53.—HUNTINGTON MILLING MACHINE. Perspective View.

The pan has a flange, L^1, projecting inwardly a short distance above the screen-openings, and the effect of this flange is to turn the material downward and prevent the tendency to rise to the top of the centrifugal and upward action as it escapes from between the rollers and the dies. Scrapers are used in connection with these rollers. O are the vertical rods or arms to which they are secured, and which are made adjustable up and down in the driving disc, J. P are the scrapers, which are made with double edges, as shown, and have two bolt-holes through them, through which bolts, Q, pass, and by which they are firmly secured to the arms.

Whenever the lower edge of the scraper becomes worn, or for any other reason it may become necessary, it is easily removed and reversed, so that the other edge projects downwards and may be made to work.

CHAPTER V.

TREATMENT OF GOLD-BEARING ORES: CONCENTRATION.

CONCENTRATION EXPLAINED—When Concentration precedes Amalgamation—Operations in Concentration—Spitzkasten: Rittenger's Pointed Box—The Trichter Apparatus—Concentration in Sluices—Further Concentration in Rockers and Buddles—Green's Jigger—The Dolly Tub—The Round Buddle—Collom's Buddle—The Concave Buddle—The Tossing Tub—Hendy's Concentrator—The Frue Concentrator—Its Operation described—Concentration of the Sulphurets—Colonel Taylor's Experience.

IN every ore there are two separate and distinct portions—the worthless and the valuable. That the valuable portion is the smaller is obvious, and in treating ores on a large scale it is an economical question how to remove the worthless portion by cheap mechanical appliances.

The previous chapters on milling ores will have shown that the whole process of amalgamating is simply a concentration of the "free gold" aided and assisted by mercury; but when gold is associated with other ingredients, concentration by means of mercury cannot be effected, and mechanical appliances are used whereby the ore particles are separated from the worthless stuff; and after the separation the valuable material is subjected to metallurgical treatment. A science has sprung up in this branch of metallurgy, called concentration, or ore-dressing. Not alone has inventive skill brought this department to such perfection as to effect the separation of the ore from the gangue, but in case of complex ores—which, for instance, carry copper pyrites, blende, and galena intimately associated together—they can be separated cleanly enough one from the other and made to yield in separate departments afterwards the copper, zinc, and lead.

The conditions which are most favourable for concentration are those where the valuable portion of the ore possesses a much greater specific gravity than the gangue accompanying it, so that in a moving body of water the heavier portion will settle and the lighter be carried away.

The concentration of fine sands and slimes—a process upon which, as I have already remarked (p. 98), depends the successful working of a gold mine—is generally performed on wooden, canvas, or metal inclined planes. The principle on which the separation of assorted grains on an inclined plane is based consists in the resistance offered by the grains, by sliding or rolling friction, to the impulse of the water on their surface.

The sorting of the sands is best performed by their free fall in moving water; and by sorting in moving or standing water different sizes of different density (that is, equal-falling grains) are obtained. Equal-falling grains are such as sink with equal speed, or which fall the same distance in the same time. Such grains do not permit of separation under water. It is, therefore, important to convey a thin layer of diluted stuff on the table. In this case the water will not strike all points of each grain with equal force, as it would in a deeper stream; but the larger, lighter particles will suffer a stronger impact of water on their higher points than the smaller ore grains, for the obvious reason that in a thin water stratum immediately on the table-plane the water has less speed, on account of adhesion, than in the upper layer. Consequently, of the equal-falling grains, with a certain medium speed of the water—depending on the inclination of the table—the larger gangue particles will be washed off, while the smaller ore grains still remain on the plane. Most tables on which the concentration is effected receive an inclination of from 6 to 8 degrees.

Although sorting is a condition of proper separation, still a perfect concentration cannot be expected, not only because of the impossibility of ever obtaining a uniform size, but also on account of difference in shape, which tends to modify the influence of specific gravity. For this reason the best concentrating apparatus will yield a medium quality stuff between the rich and the waste portions which, while not good enough to warrant extraction of the metal, will yet be too rich to throw away. The treatment of this stuff may be profitable in one place but not in another—like California or Nevada, for instance—unless it is effected in a simple way by machinery. This is now done by small elevating wheels, 4 or 5 ft. in diameter, by which this medium stuff is conveyed back to the table, so that by this arrangement only two sorts come from the concentrator—namely, the rich portion and the worthless tailings. But, notwithstanding these contrivances, it is in many cases advisable to regulate the concentration for only two educts, and to dispense with the medium stuff by turning it either into the rich portion or into the tailings, as desired.

The influence of specific gravity decreases with the increase of the fineness of the ore particles, so that in the condition of the finest slime the influence of specific gravity is very small. Hence arises the difficulty of concentrating such stuff.

Sometimes concentration of gold ores should precede amalgamation: for instance, (1) when the ore is poor; (2) when the amalgamation of middle-class ore is performed in grinding pans; (3) when the ore contains auriferous sulphurets.*

On the other hand there are cases when concentration should follow amalgamation: as (1) when ore rich in free gold is worked in grinding pans or other amalgamators; (2) when middle-class ore is treated by amalgamation without grinding.

The sands, therefore, which flow from the amalgamated copper plates are subjected to concentration to collect the free gold with the auriferous sulphurets they contain.

Concentration consists of the following operations:—

(1.) Sizing the sands by means of pointed boxes (Spitzkasten).

(2.) Concentrating the sands in the pointed boxes in sluices, having a self-raising gate or riffles.

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* See my "Metallurgy of Silver," *art.* Milling at the Montana Mine.

(3.) Subjecting the sands in the sluice to a further concentration in rockers, buddles, Hendy's concentrators, &c.

(4.) Giving the buddle concentrates a more perfect cleaning in the tossing tub.

(5.) In more modern mills, the pulp from the battery discharges directly on to vanners, or concentrating tables, for concentrating the escaping gold and sulphurets, or they discharge first into sizers and from these on to vanners.

(6.) In some establishments where the battery pulp is subjected to pan amalgamation, the tailings from the settlers are discharged on to vanners, and to concentrate the resulting finely-ground slimes it is advisable to discharge from the vanners on to round buddles, which prove effective for treatment of very finely-ground material.

Spitzkasten.—Rittenger's Funnel or Pointed Box.—In this apparatus the funnel boxes are rectangular pyramids, with the base upwards. They were designed to replace the "labyrinths" or settling pits—the oldest classifying arrangement—for assorting sands for concentration directly from the battery. The battery sands are too fine for sorting by sieves.

The stuff flows from the battery through several of these boxes—each different in size, and each delivering a different sized grain—directly to the concentrators. In this way the sands may be separated into different degrees of fineness at very little expense, in a simple apparatus, and without the help of hands.

These boxes have also the advantage of getting rid of incurring surplus water,. which would interfere with concentration. The sands are always obtained from the boxes in the necessary state of dilution for the tables. The principle is the same as with the labyrinth. The first box is narrow, which causes the water to flow swiftly, thus allowing only the coarser grains to sink, while the balance is carried into the next wider box, where the same quantity of water, spreading, assumes a slower motion, so that the next finer sand is separated, and so on with the other boxes.

Figs. 54 and 55 represent a pointed box; Fig. 54 is a vertical longitudinal section, Fig. 55 a vertical cross section. The stuff flows from the battery, or from a sifting apparatus, at *a*, into the box. The sinking grains concentrate at the point *o*, and flow out through the ascending conduit, *o′*, into the trough, *p*. The finer stuff which could not resist the current of the water is carried over the spout, *b*, into the next larger box. The conduit, *o′*, ascends in order to counterbalance the water pressure inside the box.

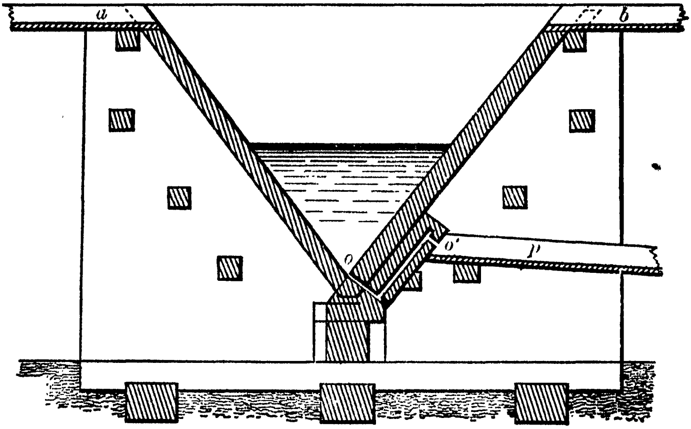

FIG. 54.—SPITZKASTEN. Vertical Longitudinal Section. Scale ½ in. = 1 ft.

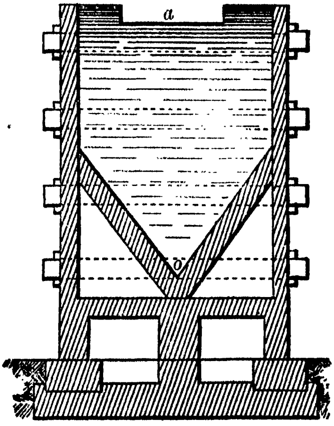

FIG. 55.—SPITZKASTEN. Vertical Cross Section. Scale ½ in. = 1 ft.

To obtain all the advantages of this apparatus theoretical and practical knowledge must be taken account of in construction. The width of the separate boxes is important. It depends on the quantity of stuff which is intended to enter the box in a second, and on the degree of the coarseness and density of the grains in it. According to experience, the first box by which the coarsest part of coarse ore is to be separated should receive one-tenth of a foot width to each cubic foot of stuff flowing per minute. Each of the next following three boxes receive twice the width of the preceding.

If, therefore, such an arrangement be intended to separate 10 cubic ft. of coarse ore per second, the first box should receive 10 × 1·10 = 1 ft. width, and the whole four boxes as follows :—

1, 2, 4, 8 feet width, and
3, 6, 9, 12 ,, length.

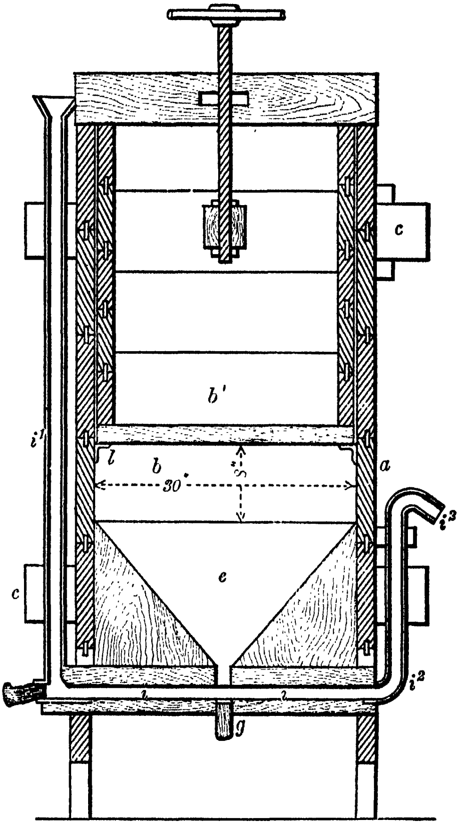

FIG. 56.—SPITZ LUTE. Scale ½ in. = 1 ft.

The depth is given by the inclination, and measured from a horizontal line is fifty degrees. This inclination is given to the sides of the width. The conduit, *o'*, can be formed also by a pipe, *i*, Fig. 56, which stands vertical on the last boxes, containing the fine sands. The pipe should be shorter with coarser sands.

Should it be necessary to obtain a thicker stuff for the concentrator, there must be a valve at *o*, opening at short intervals. The connecting troughs between two boxes (Fig. 57) widen towards the larger boxes, and must be sufficiently inclined to insure that no sand can deposit in them. The coarser the sand, the more inclination. The boxes are not always close together, in which case the conveying troughs are narrow, and discharge in a distributing board of the next box.

The conveying troughs have a square section. It is calculated that five square inches will answer for each cubic foot of stuff per minute. If, for instance, seven cubic feet of diluted stuff flows through the trough per minute, it must have $7 \times 5 = 35$ square inches or about 6 by 6 in the clear.

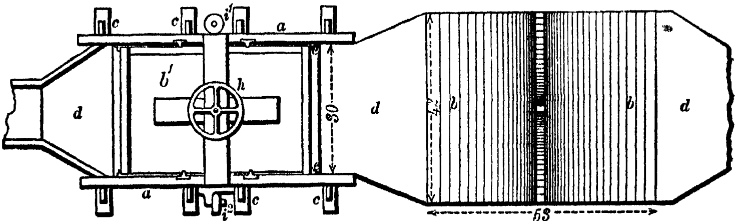

FIG. 57.—SPITZ LUTE. Scale ¼ in. = 1 ft.

The inclination of the conveying trough must be at least, in each 6 ft.—

For coarse sands	1 to 1½ inch.
,, middle fine	½ ,
,, fine	¼ ,
,, slime	⅛ ,

This inclination is for galena ore containing not over 5 per cent. of galena. The outlet of the pipe *o'* (Fig. 54) is regulated by mouthpieces, of which there must be several of different sizes. Once these are regulated, there is no other work to be done, except to watch the regular flow, as some accident may cause the conduit to become choked. To obviate such mishaps the larger boxes are provided with a rod in the centre, on the lower end of which, just above the opening, are two small rings, so that if the sand or pulp accumulate by neglect of the watchman this rod is turned unti the stuff becomes loose enough to force itself through.

The percentage received from boxes of the dimensions described is as follows:—

From the first box . .	40 per cent. of sand.
,, ,, second ,, . .	28 ,, ,,
,, ,, third ,, . .	18 ,, ,,
,, ,, fourth ,, . .	10 ,, ,,

So that the loss is not more than from 4 to 6 per cent. It sometimes occurs that several stamps have to stop working, in which case the supply of crushed stuff is diminished. It is then necessary to replace this deficiency of stuff by water. But if the supply of water depends on circumstances that cannot be regulated, or if the deficiency cannot be counterbalanced by water, then other means must keep the fluid in the normal condition, and for this purpose each box is provided with a movable longitudinal board. It is sufficient to make this board from 12 to 18 in. wide. In proportion as the supply of water is less, the dividing board is moved towards one side. If, for instance, the supply should decrease to half the normal quantity, the board is placed in the middle of the box, and all the stuff must enter on one side. The active portion of the water reaches only a few inches below the surface, so that the dividing board need not touch the bottom.

In Schemnitz (Hungary), for crushing 20 tons in twenty-four hours, the boxes have the following dimensions * :—

First box . .	6 ft. long,	2¾ ft. wide,	4 ft. deep.
Second ,, . .	9 ,, ,,	5 ,, ,,	6 ,, ,,
Third ,, . .	12 ,, ,,	9 ,, ,,	8 ,, ,,
Fourth ,, . .	16 ,, ,,	15 ,, ,,	10 ,, ,,
The first box gives . .	40 per cent. of sand.		
,, second ,, ,, . .	22 ,, ,,		
,, third ,, ,, . .	20 ,, ,,		
,, fourth ,, ,, . .	12 ,, ,,		

The Trichter Apparatus is shown in Fig. 58, which is a top view. The first two boxes, *a a*, are narrow and oblong; then follow three other boxes, *b b*, square in section, set 2 ft. 10 in. each side; next two rows of four boxes, *c c. . . . c*, in which the stuff divides. The depth in each is 20 in. By means of gates each box can be shut off if required.

The small boxes could be advantageously used for sizing the stuff or tailings of unroasted ores from the pan amalgamation if concentration were intended; but in order to obtain regular work there should be a large agitator,[*] with a continual discharge into funnels, which latter should be proportioned in number and size to the quantity of the tailings. A common receiving agitator is necessary, because the discharge of the tailings from the settlers is in most mills periodical. Where the pulp is excessively diluted in the settlers, the agitator, if not large enough, must be provided with a trough at top to carry off surplus water.

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* According to Prof. G. Kustel.

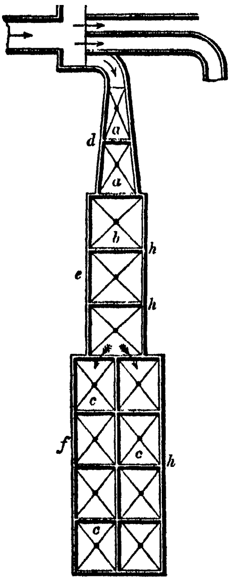

FIG. 58.—TRICHTER APPARATUS.

Concentration of the Pointed Box Sands in Sluices with Self-raising Riffle-gate.—The pointed box sands flow through wooden sluice boxes of a rectangular section, provided at the lower end with a self-raising gate, acting as a riffle, in which the heavier portions of the sands, consisting of sulphurets, black sand, &c., form a deposit near the head, while the lighter particles escape over the gate. For the coarser sands the boxes have a width of 1 in.; for the finer sands they are from 1⅛ to 1½ in. wide for every stamp. They are usually 18 ft. long, and have the natural grade for the passing of the sands through them. For every two boxes or two sets of boxes there is one riffle-gate. The gate opening is generally 7 in. wide and 18 in. deep. The gate is raised to its full height once in twenty-four hours by means of a horizontal ratchet wheel, which is keyed to a vertical screw attached to the gate. Two ratchet wheels, one above the other, are attached to the screw, the teeth of one having the reversed direction of those of the other. By applying a lever arrangement on one side of the vertical screw to one of these the gate is raised; by applying it to the other on the other side the gate is lowered. Two boxes, 15 to 16 in. wide, are filled to a depth of 18 in. by a 15-stamp battery in twenty-four hours. Two boxes or two sets of boxes are used alternately.

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[*] See my "Metallurgy of Silver," *art.* Working Tailings at the Tombstone Mines in Arizona.

Concentration in Rockers and Buddles.—When the sluice sands are subjected to a further concentration on rockers they are discharged into a tank. The tailings of the rocker containing the finer sulphurets are treated subsequently in buddles, worked either by horse or by steam power.

The Rocker.—The rocker, represented isometrically in Fig. 59, consists of a wooden table of 2-in. pine plank, 20 in. wide, and 10 ft. long, supported at both ends by wooden rockers, representing a section 20 in. in width and about 3 in. in depth. The table is enclosed on the long sides and the upper end by 6-in. boards, the lower end being left open for the discharge of the tailings. It has an inclination of 1 in. to the foot. This inclination can be increased, when working coarser sands, by removing some of the supporting scantlings at the lower end. The upper end is fixed by a bolt, which, working in a slot, does not prevent the rocking motion. The floor of the table when in equilibrium is 7 in. above the support.

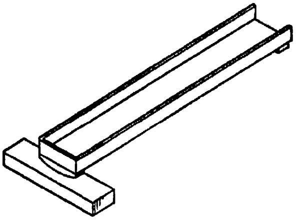


FIG. 59.—THE ROCKER.

The concentrated sluice sands are introduced at the head of the rocker in charges of from 3 to 5 shovels (the greater quantity corresponding to the sands poorest in sulphurets), and a stream of water discharged by a 1-in. pipe under a 6-in. pressure is turned on the sands by means of a rubber hose. The rocker is set in motion by the left hand of the workman, giving it about 60 strokes of 8 in. a minute. For coarser sands a greater number of strokes is required. The lighter sands gradually work down, while the sulphurets remain nearer

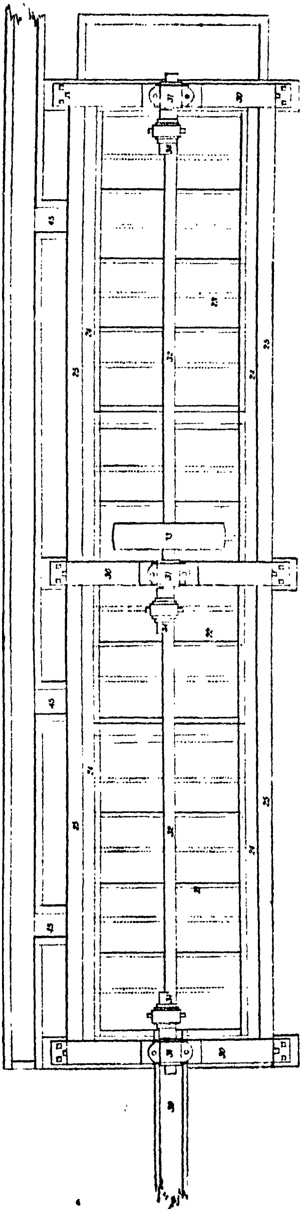


Fig. 60.—Green's Jigger. Plan.

the head. With his right hand the workman works the sulphurets up to near the head with a flat wooden shovel, which passes closely along the bottom of the rocker, while the lighter sands pass off. In some mills the rocker receives its motion by machinery. When clean the sulphurets are removed from the rocker by an iron scoop. A workman can treat 300 shovels of sand in a ten-hour shift.

Simple as this machine may appear, it does very good work, when operated with a proper quantity of water, right motion, and exact inclination.

Green's Jigger (Figs. 60, 61) consists of a series of sieves, which receive a reciprocating movement up and down in water by means of eccentric or revolving cranks. The space below is portioned off, so that each sieve discharges into a separate compartment. Not alone does this apparatus effect a concentration of the ore, but by making the sieves of different degrees of fineness a sizing can be effected. Fig. 60 is a plan, and Figs. 61 and 62 vertical sections, of the apparatus as fitted with three sieves, 21, 22, 23. These are fixed in a rectangular frame, 24, fitted to work up and down in a fixed rectangular frame, 25, forming the upper part of a tank or hutch, which is supplied with clean water by a pipe, 26. The bottom of the tank is formed by three inverted pyramidal cast-iron shells, 27, 28, 29, forming separate compartments respectively under the sieves 21, 22, 23, and the whole is supported by cast-iron standards, 30.

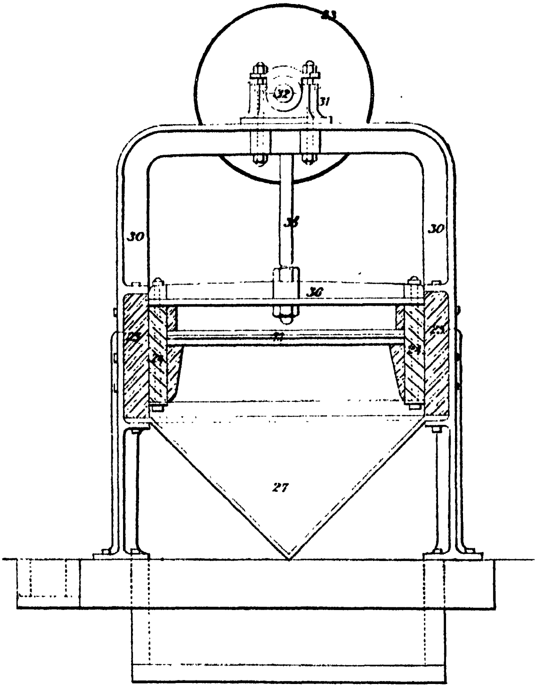


Fig. 61.—Section of the Same.

These standards are fitted with bearings, 31, for a horizontal shaft, 32, running along above the middle of the tank, and driven by a belt acting on a pulley, 33. The shaft, 32, is formed with three cranks, 34, of a small throw, to which are adapted brasses working in transverse slots formed in the heads of three connecting rods, 35, fixed to the cross bars, 36, of the

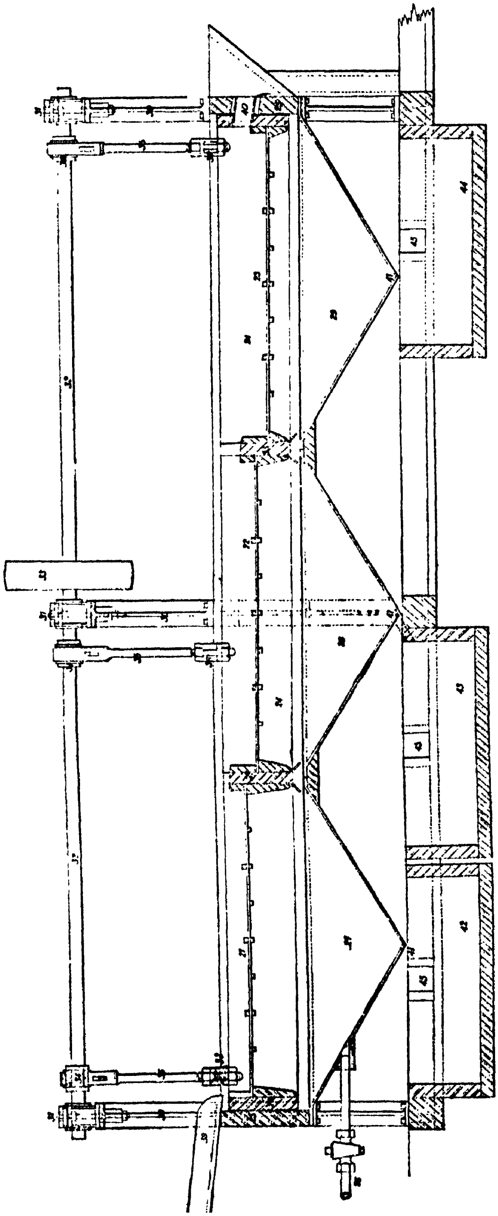


Fig. 62.—Longitudinal Section of Green's Jigger.

sieve frame, 24. The sieves, 21, 22, 23, are fixed in the frame at successively lower levels, and there are transverse division pieces, 37, 38, between the sieves, with their upper edges cut down to the proper overflow level for each sieve.

The stuff to be acted upon is fed by water from a launder, 39, upon the highest sieve, 21, and what is not separated by this sieve overflows to the middle one, 22, and some also overflows from the middle one to the third, 23. The waste finally overflows by a discharge duct, 40, at the end of the third sieve. The materials which pass through the sieves may be discharged at intervals from the compartments below, 27, 28, 29, by opening valves or sluices; or a continuous discharge may be effected into the receptacles 42, 43, and 44, these receptacles being formed with overflow ducts, 45, for the water to pass off by.

The Dolly Tub.—This is an apparatus used for separating ores, which can be conveniently used with the separators or classifiers, and will concentrate very fine slimes after they leave the spitzkasten or any similar apparatus. It is shown in Figs. 63 and 64 in plan and section.

It consists of a cylindrical vessel, 1, of cast-iron or wood, and has a raised centre, 2. Inside the tub revolve the paddles, 3, carried by the arms, 4, which are fastened to a vertical shaft, 5, held in bearings in a framing, 6, fixed across the top of the tub. The vertical shaft, 5, is geared by a pair of toothed bevel wheels, 7, to a horizontal shaft, 8, also carried by the framing, 6, and which has on it the fast and loose pulleys, 9, to receive a driving strap.

The part, 10, of the bottom of the tub between the centre, 2, and the sides, 1, is by preference sloped downwards from the centre or made conical, as shown. A funnel, 11, is fitted to the top of the raised centre, 2, and leads to a central discharge pipe, 12, by which the water and the light slime or waste is discharged.

When the apparatus is used in combination with "classifiers," each class of materials is fed to a different dolly tub or at a different time from the same tub. The materials, along with a regulated supply of water, are led by a launder so as to fall into the dolly tub, by preference near the side, and the whirling motion produced by the revolving paddles, 3, and the consequent centrifugal tendency cause the heavier minerals to move to the side, and they gradually descend to the deepest part of the bottom, from a point in which they are withdrawn by an aperture, 13, of a regulated size.

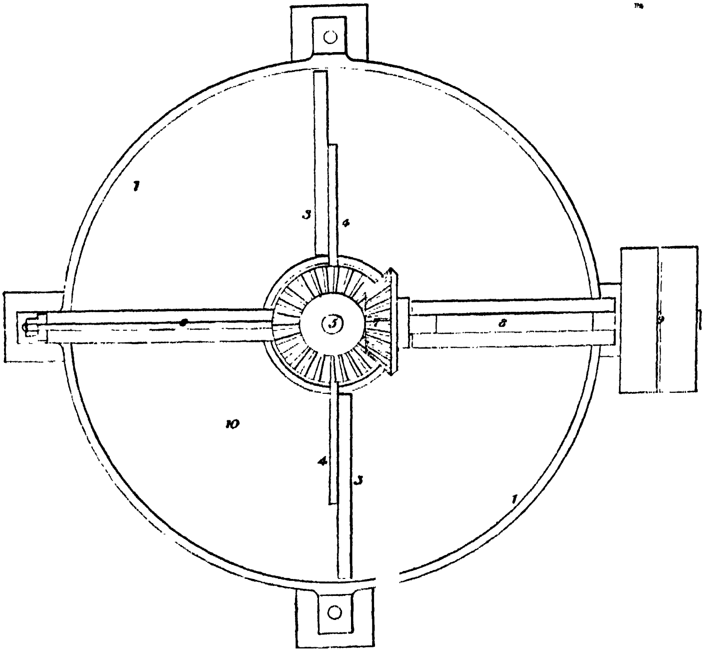


FIG. 63.—PLAN OF DOLLY TUB.

In some cases the revolving paddles, 3, may be dispensed with, and the desired action be obtained by directing the inflowing water tangentially, or otherwise, so as to produce a suitable motion; and the action may be controlled by regulating the ingress of water, whilst the egress is of course adjusted to maintain a suitable depth of water in the vessel.

The whole of the water may be supplied along with the slimes, or a portion of it may be introduced by one or more

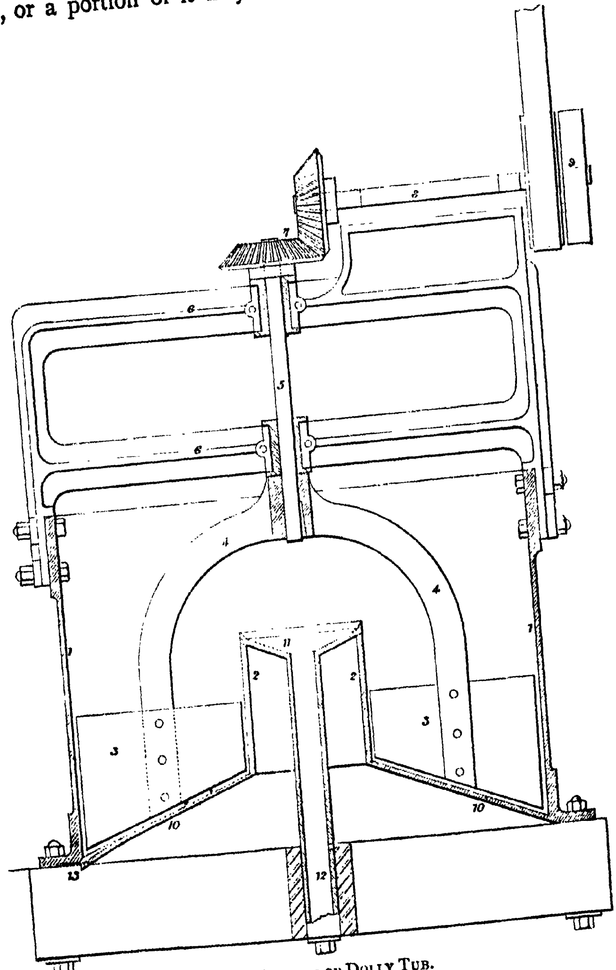


FIG. 64—SECTION OF DOLLY TUB.

separate pipes. The discharge of the water being by the central funnel, 11, there is a continuous movement throughout the water towards that point. The circular motion, however, produces an opposite or centrifugal tendency, which, acting principally on the heavier particles, carries them to the sides of the vessel, whilst the lighter particles, being more under the influence of the inward current, are by it carried to the discharge pipe, 12. The supply and discharge to and from the apparatus may be either continuous or intermittent.

The Round Buddle.—This machine serves to concentrate slimes and fine sediments on a circular bottom, inclined toward the periphery. It is represented in Figs. 65 and 66. The conical bottom, *a*, is formed of wood, and is 16 ft. in diameter, or sometimes 20 ft. On this the stuff is distributed. *b* is the cone supporting the upper part of the feeding apparatus; *e*, a funnel perforated with four holes, and furnished at the top with an annular trough; *f f* are arms, carrying two brushes, balanced by the weights, *g g*; *h* is a launder for conducting the stuff into the funnel, *e*, from which it passes through the perforations, flows over the surface of the fixed cone, *b*, and from thence towards the circumference, leaving in its progress the heavier portions of its constituents, while the surface is

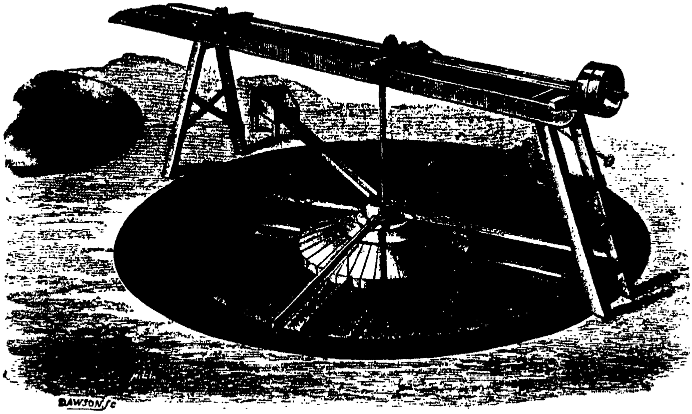


FIG. 65.—THE ROUND BUDDLE.

constantly swept smooth by means of the revolving brushes. By this means the particles of different densities will be found arranged in concentric circles.

The arms usually make from two and a half to four revolutions per minute, and a machine 18 ft. in diameter will work up from fifteen to twenty tons in ten hours.

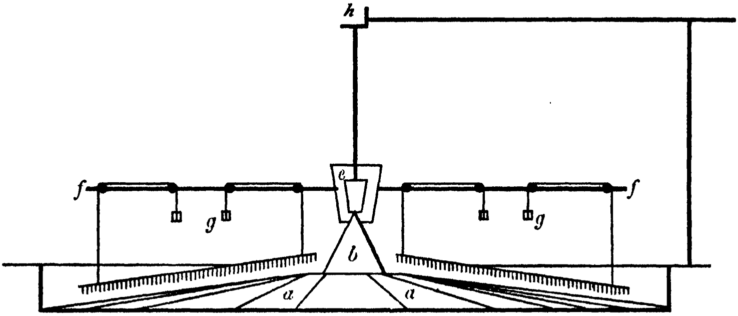


FIG. 66.—THE ROUND BUDDLE. Scale $\frac{3}{16}$ in. = 1 ft.

Collom's Buddle differs from the above by dividing the revolving table into two parts or concentric circles, one part being arranged by preference at one angle, whilst the other is arranged at a different angle. A pipe for washing off the ore is so formed as to wash off the ore from the different portions of the table at separate points, the ore from the upper portion of the table being washed off by one portion of the washing-off pipe into a launder or shoot and carried thereby into a receiver or hutch without touching the lower portion of the table, whilst the ore from the lower portion of the table will be afterwards washed by another portion of the washing-off pipe into another receiver or hutch. The water and the refuse or tailings from the general surface of the table will be washed into circular receivers or hutches. The table may be divided into two or more parts or concentric circles arranged at the same or different angles, and may be provided with several washing-off tubes, which may be arranged to operate alternately or together. The ore is fed to the apparatus by means of a launder or shoot through which a stream of water flows, and the ore is washed, dressed, or cleaned by means of a continuous flow of water from a circular tube.

This buddle is applicable to the concentration or dressing of both gold and silver ores, and consists of a circular table in which there are grooves or channels for the reception of mercury. These grooves are arranged in concentric circles, and as the pulp passes over them the mercury absorbs any amalgamable gold passing over it; while the heavier ore particles settle on the table, the remainder will be washed off by jets of water from washing-off tubes into suitable hutches or receivers. The

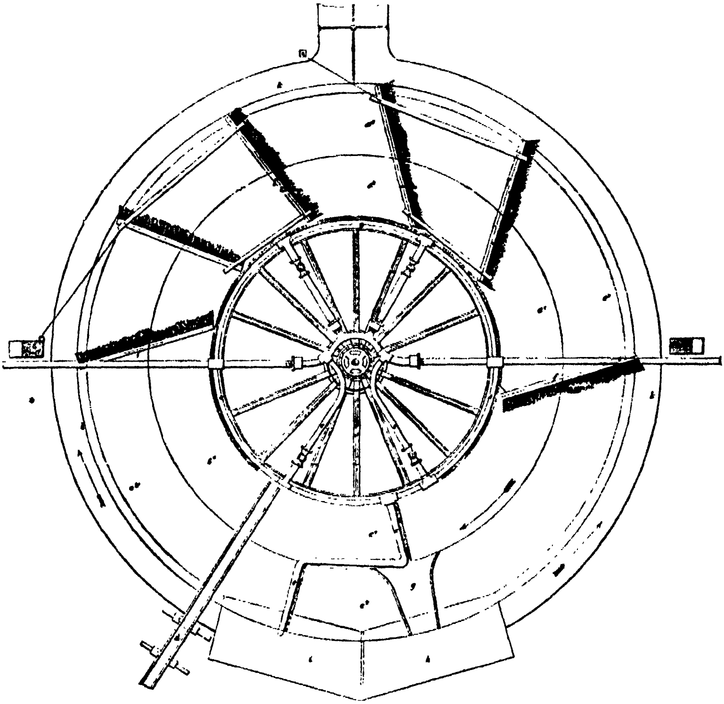


Fig. 67.—Plan of Collom's Buddle.

water, and the refuse or tailings from the general surface of the table will be washed into circular receivers or hutches. The tables forming the different concentric circles are generally set at different angles. The tables may be arranged to incline to or from the centre.

A clearer understanding of the machine will be obtained from the accompanying illustrations.

Fig. 67 is a plan of the machine; Fig. 68 is a vertical section of the same. *a*, *a*, is a vertical axis having a bevelled wheel at its upper end gearing with a similar on an horizontal shaft, *b*, which is driven in any convenient manner, and in this way the axis, *a*, is caused to revolve at a speed of about two or three revolutions per minute. On the axis, *a*, a table is mounted; it may be conveniently about 25 ft. in diameter, and this table consists of two rings, c^1 and c^2, or it may be of three or more rings if it be desired to sort the ore or material into a greater number of qualities. *d* is a trough or launder through which the ore or material in a finely divided

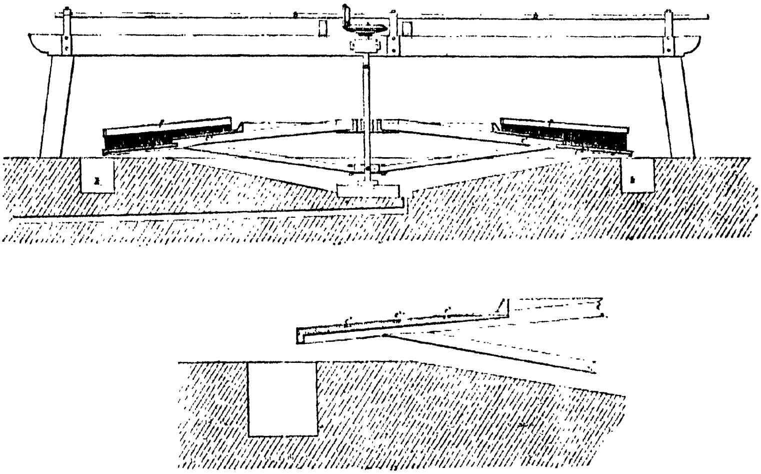


Fig. 68.—Section of Collom's Buddle.

state and mixed with water is run on to the inner side of the smaller ring, c^1, of the table; e, e, are perforated pipes from which jets of water issue, these pipes (it will be seen) being omitted in Fig. 68.

As the table revolves, the ore upon it is subjected to these jets, and the particles are carried a greater or less distance down the table according to their relative densities and the strength of the jets of water, which can be regulated by stop-cocks in such a manner that as the table moves round the material upon it becomes exposed to stronger and stronger jets, but yet the jets should not be so strong as to wash any considerable proportion of the more valuable material entirely off the lower ring, c^2, of the table; f, f, are brushes resting loosely on the table and serving to move and distribute the material. The brushes are held and kept in their places by cords or light rods; they should not bear so heavily on the table as to cause the material to accumulate against them.

In the course of the revolutions of the table, all but the heaviest and best material is washed off the upper and inner ring, c^1, of the table on to the lower and larger ring, c^2, the slope of which is by preference somewhat steeper than that of the upper ring. Here a second sorting takes place, the lighter parts being washed over the edge of the table, whilst those which are heavier remain, being able to resist the flow of the water; $e^{\times}$, $e^{\times\times}$, are perforated pipes throwing jets of water, and so arranged and adjusted as to wash all the material from the table as it passes beneath them. When the material arrives opposite to these pipes it is already sorted. The jets from the pipes $e^{\times}$ wash down the best quality into the trough or launder, g, which conveys it into a receiver h, and the jets from the pipes $e^{\times\times}$ wash down the material of second quality from the lower and outer ring c^2 into the receiver i. k is a channel all round the table, which leads off the water and the lighter material to settling points.

The lower portion of Fig. 68 shows a transverse section of a portion of a table, which is grooved all round for receiving mercury. The grooves, which are marked $c^{\times}$, may conveniently be three in number, and each about three inches wide by half an inch deep. The mercury with which they are filled takes up particles of gold or silver, as already stated; and when it becomes fully charged, which will only be at long intervals, it is drawn off and renewed.

The Concave Buddle.—The concave buddle of Paine and Stevens is shown in Fig. 69. There are generally two buddles: one for the coarse sluice concentrates, and the other for the finer. They have an exterior diameter of 18 to 20 ft., slightly inclined inwards, in the centre of which is an opening 2½ ft. in diameter. The vertical shaft is supported by the wooden block, *m*, which carries the journal box. The shaft has a rotary motion imparted by mill-gearing above, and to which several appendages are attached.

Attached to the shaft are: (1) The self-raising riffle pulley, *g*, which is raised by a rod, *p*, receiving its upward and downward motion from the endless screw, *b*, and pinion wheel; (2) the arms, *f f*, carrying the brusher; (3) the sand distributing troughs, *e e*. The clear water box, *i*, is suspended by the wheels, *v v*, on an annular flat ring. It is supplied by the stationary wooden box, *r*, and discharges the water, by the iron pipe, *k*, into the sieve boxes, *y* and *z*. The bore, *s s*, is fed by the trough, *h*, from the mixing trough, *y*, and sieve box, *z*. The vertical shaft receives its motion by the pulley, *a*, and bevel gearing, *d d*.

The first operation of the buddle consists in washing the sluice concentrates. For this purpose the sluice gate is lowered and the sands are gradually washed through the mixing and sieve box, *y* and *z*, and box, *h*, into the distributing box, *s s*, from which they are discharged by the six revolving Russia iron trough arms, *c c*, upon the annular apron. From these they flow down the inclined conical table, constituting the buddle. The arms, *f f*, carrying the poles to which the brooms are attached, revolve at the same time with the distributors, *c c*. The brooms maintain a regular and even surface to the sands. The heavy sulphates lodge near the head of the buddle, while the lighter particles move on and finally fall over the circular riffle, *g*, which consists of an iron pulley in the centre of the

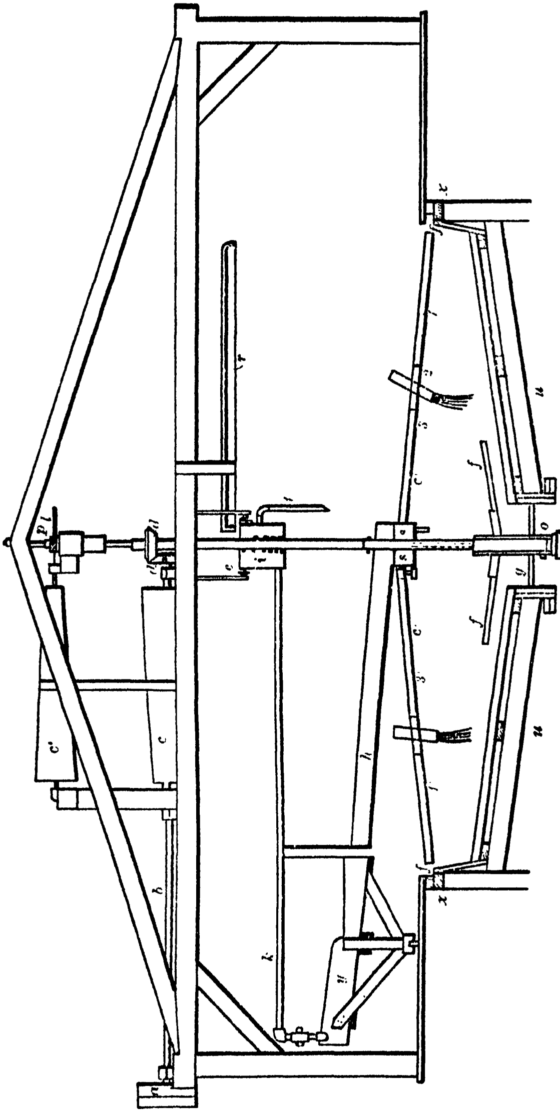


FIG. 69.—THE CONCAVE BUDDLE. Scale 1 in = 5 ft.

buddle. This ring forms a close joint at the end of the wooden buddle floor, *u u*. The pulley, and with it the arms of the broom poles, are gradually raised as the washed sands fill the buddle.

When working the coarser ore, the time consumed to fill the buddle is six hours; for the finer sands twelve hours are required. The water required by the buddle in washing the coarse sands is 8 cubic ft. per minute, spread by the revolving distributors, which make seven revolutions a minute over a periphery of nearly 63 ft.

When the buddle is full the sands are divided into three concentric rings. The interior ring, 3½ ft. in width, is considered as waste, and is washed out. This is done by removing the end pieces, 1 1, of the distributors, and turning the collars, 2 2, in the manner represented in the drawing by dotted lines. The clear water is conveyed from the stationary sluice, *r*, and washes out the central ring, leaving the distributors at the connecting collars, 2 2. The central ring is lowered at the same time. While the central ring of the sands is removed the outside ring at the periphery of the buddle, 3 ft. in width—the so-called headings—is shovelled out by the ore dresser. The middle portion of the sands, being about 2 ft. in width, is gradually washed with water directed by means of the collars, 2 2, of the distributors. The headings of these sands are added, when washed, to the first headings, and the central ring of sands is removed as we have described.

The headings from the first washing go through a second washing, called doubling. This is done by raising the central riffle ring about three inches and filling the buddle up to the rim of the ring with poor sands from the tossing tub, which are introduced into the mixing box, *y*, and washed into the buddle by the water from the pipe, *k*, and trough, *r*. When ready for doubling the buddle has an inclination of 1¾ in. per ft. for coarse sands and 1½ in. for the finer sands. All the headings are shovelled into the mixing trough, which, being supported on wheels, can be brought to any part of the working floor around the periphery of the buddle. The distributing clean-water box, *i*, with water pipe, *k*, attached, also revolves on the wheels, *u u*. The time consumed in re-washing the headings of the first washing is generally three hours.

When the operation of washing the headings is completed the sands are again divided into three concentric rings. The central portion, having a width of about 4 ft., being removed as before, the outer ring, about 18 in. in width, is ready for tossing, while the middle ring, of about 3 ft. in width, is washed down as described, the heading of it being added to those previously removed by tossing.

The Tossing Tub.—The tossing, or final cleaning of the sulphurets, if they are to be treated by the chlorination process, is usually performed on the buddle headings. This is done in a tub of the following description (Fig. 70). The tub, having 1½ to 2 in. staves, is conical in form, tapering toward the bottom, 4 ft. in diameter, and 2½ ft. deep in the clear. Through the axis of the tub a hollow cast-iron cone, *c*, passes, reaching a few inches above the top of the tub, and fastened by a flange to the bottom. A shaft, *s*, Fig. 71, passing through this cone and resting on a journal underneath, carries the yoke, *l*, to which the horizontal flat-iron stirrers are riveted. Motion is communicated to the shaft by bevel

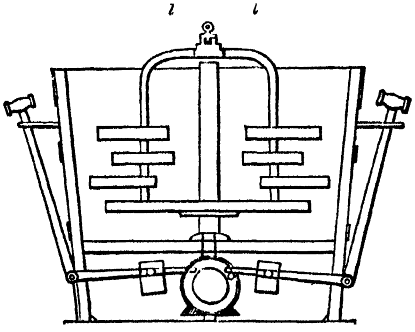


FIG. 70A.—THE TOSSING TUB.

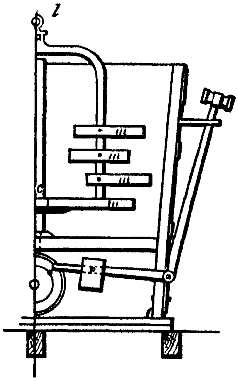


FIG. 70B.

Scale $\frac{5}{16}$ in. = 1 foot.

gearing, *a b*. The hammers are set in motion by the pins, *r r*, attached to the vertical bevel gear, as seen in the figure. When ready for tossing, the tub is filled to nearly half its height with water, the stirrers are set in motion, making forty-eight revolutions a minute, and the ore is shovelled in near the periphery of the tub. When nearly full the yoke is lifted out by means of a rope and pulley overhead, and the sands are allowed to settle while the hammers are set in motion, making ninety-six strokes each per minute to facilitate the rapid settling of the sulphurets and sands. When the sands have settled, the water is drawn off by an iron syphon, the skimmings are removed to a depth of two inches and thrown out as waste, the remaining upper half of the sands are re-tossed, and the resulting sands above the sulphurets washed again in the buddle during doubling. The lower half of about five to six inches, consisting of sulphurets sufficiently concentrated, is delivered at the chlorination works to be further treated for gold.

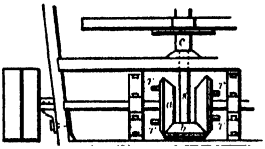
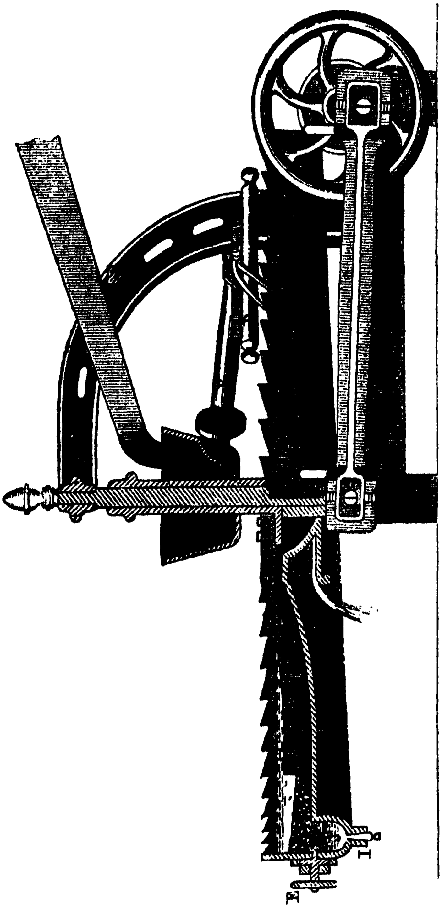


FIG. 71.

Two buddles working the sands of a 30-stamp mill require the attendance of three men.

The riffle boxes and, to some extent, the concave buddles, lose a considerable proportion of coarse sulphurets. To minimise this loss, the tailings from the buddles and riffle sluices pass through a long string of boxes (supplied with wooden riffles of a square section of $\frac{3}{4}$ of an in., which are washed every Sunday when the mill stops) into and through a wooden box about 5 ft. wide, 10 ft. long, and 3 ft. deep. This box is open at the lower end. The heavier sulphurets form a deposit at the head, and the headings are treated by the rocker.

In concentrating by means of riffle sluices, concave buddles, and rockers for buddle tailings, the loss in sulphurets is 12·5 per cent. when treating quartz sands, which contain about ·33 per cent. of sulphur, or its equivalent in sulphurets.



HENDY'S CONCENTRATOR.

Hendy's Concentrator.—When this concentrator is employed, it is not usually necessary to use the pointed box, nor previously to concentrate by self-raising riffle boxes. The concentrator is shown in Fig. 72, and consists of a shallow pan 5 or 6 ft. in diameter, supported by a vertical shaft in the centre, and made to oscillate by cranks on one side; these are joined by connecting rods with the periphery of the pan, and this turns upon a vertical axis through a short distance for every revolution of the crank shaft. The bottom of the pan is raised in the centre to nearly the height of the rim, in order to facilitate the movement of particles towards the circumference.

The machine must be carefully levelled; and the tailings, direct from the blankets, are delivered by a trough to the hopper, *c*, from which they pass through the pipe, *k*, and distributor, *d*, into the pan near its outer edge; and the rotation of *d* by means of two pawls attached to it, and acted on by teeth on the rim of the pan, causes this delivery of the tailings to take place evenly at all parts of the circumference. Rake-like arms, *m*, rotate with *d*, in order to stir the compact mass of sand, &c., which settles at the bottom. The crank makes 210 revolutions per minute, and the accumulated sulphurets are discharged through the gate, *e*, while the amalgam and mercury collect in the depression, *j*. The machine concentrates 5 tons of tailings per twenty-four hours.

Mill with Hendy's Concentrators.—Fig. 73 shows the general arrangement of a gold mill, and from it a clear idea will be at once gathered of the several processes which intervene between the raising of the ore and the final escape of the waste sands, &c., into the tail sluices.

The ore is thrown down in front of a Blake rock-breaker, and after leaving it passes down an incline to the self-feeder, and thence to the stamps, where amalgamation often takes place. In front of the stamps is an apron of amalgamated plates, and beyond are the blanket tables where the pulp undergoes a mechanical preparation, the resulting two qualities being differently treated. The sands which pass over are received in Hendy's concentrators, and after being a second time concentrated pass into the tail sluices.

That which remains on the blankets is washed into tanks, then passed through Attwood's amalgamators and over copper riffles, and finally reaches the second concentrator. Below the concentrator in the figure are seen a Wheeler pan and settler. The pyrites obtained on the concentrators are usually roasted and treated by chlorination.

The Frue Ore Concentrator or Vanner.—This machine (Fig. 74), which is of American invention, has been found so efficient in the concentration of pulp and slime that I shall describe its construction and working in full detail.

Its principle is that of a revolving blanket, having a lateral shaking motion—a motion closely resembling that given to a shovel in vanning by hand. Revolving belts of canvas, with a

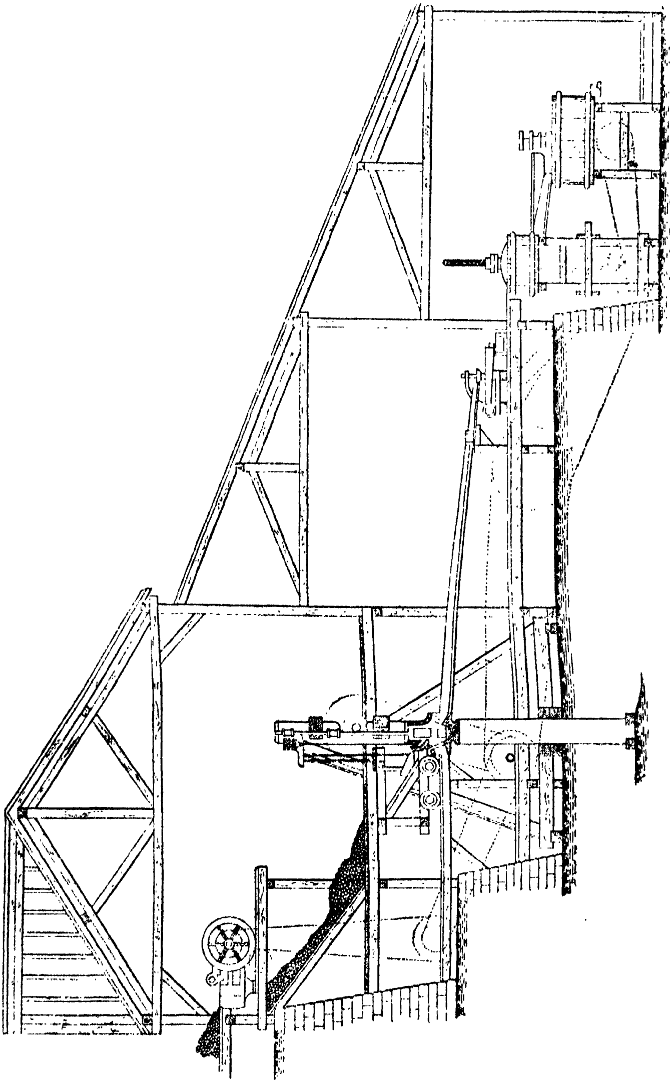


FIG. 73.—MILL WITH HENDY'S CONCENTRATORS.

lateral or end blow, have been used heretofore with pretty good results, but the side shake has proved a great improvement on the old practice, and india-rubber belts with high flanges have been introduced in place of canvas. The ore when stamped fine can be received directly from the stamps on to the vanners, which concentrate the ores to a high percentage, including those which contain galena, zinc blende, iron and copper pyrites.

In Fig. 75, A A are the main rollers that carry the belt and form the ends of the table. Each roller is 50 in. long and 13 in. in diameter; it is made of sheet-iron, galvanized, and is light and strong.

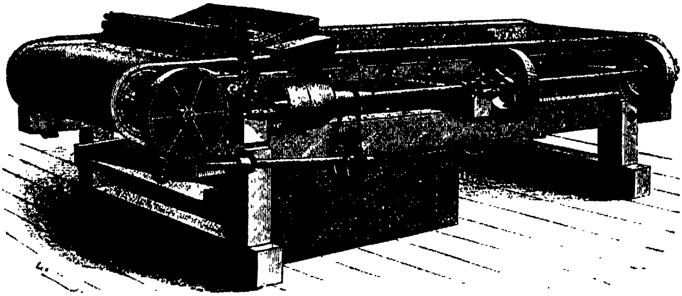


Fig. 74.—The Frue Concentrator or Vanner.

The bolts which fasten the boxes of A A to the ends of F also fasten to F the upper supports which rest on uprights, N.

The rollers, B and C, are of the same diameter, and are made in the same way as A A. The roller part of C is shorter than that of A A and B, and has also rounded edges, the upper surface of the belt with its flanges passing over it. The belt, E, passes through water underneath B, depositing its concentrations in the No. 4 box, and then, passing out of the water, the belt E passes over C, the tightener roller. B and C are hung to the shaking frame, F, by hangers, P P, which swing on the bolts fastening them to F. By means of the hand screws, B and C can be adjusted on either side, thus tightening and also controlling the belt (Figs. 76 and 77).

The boxes holding A A in place have slots and adjusting screws, so that by moving them out or in A A can be made to have a very strong influence on the belt E; and as E sometimes travels too much towards one side, this tendency can be stopped most quickly by lengthening or shortening on one side or the other of A A, remembering that the belt always travels to the near side of the pulley. The swinging of B or C also controls the belt. *c c* are bolts and washers to take up the end play of rollers, A A; the bolts pass through holes in the gudgeons of A A, Fig. 75.

D D are the small galvanized iron rollers, and their support causes the belt, E, to form the surface of the evenly inclined plane table. This moving and shaking table has a frame, F, of ash, bolted together, and having A A as its extremities. This frame is braced by five cross pieces. The bolts holding together the frame pass through the sides close to the cross pieces; the cross pieces are parallel with A A and D D, and their position can be understood by the three flat spring connections, R Q, which are bolted to three of them, one to each, underneath the frame.

The belt, E, is 4 ft. wide, 27½ ft. in entire length; being an endless belt of rubber with raised sides.

G G is the stationary frame. This is bound together by three cross timbers, which are extended on one side to support the crank shaft, H.

G G supports the whole machine. The required inclination of the table is given by elevating or depressing the lower end of G G. This is accomplished by means of wedges. The frame rests on uprights, Nos. 3 3, fastened to the sills which form the foundation of the machines in the mill.

F is supported on G G by uprights, N, four on each side. These uprights are of flat wrought iron, with cast-iron bearings above and below. Each middle bearing on F has one bolt hole; there are two of them on each side. The end bearings have two bolt holes, there being four of them, two on each side. These bolts pass through the frame, F, and also hold to the frame the bearings of A A, which work in a slot. The

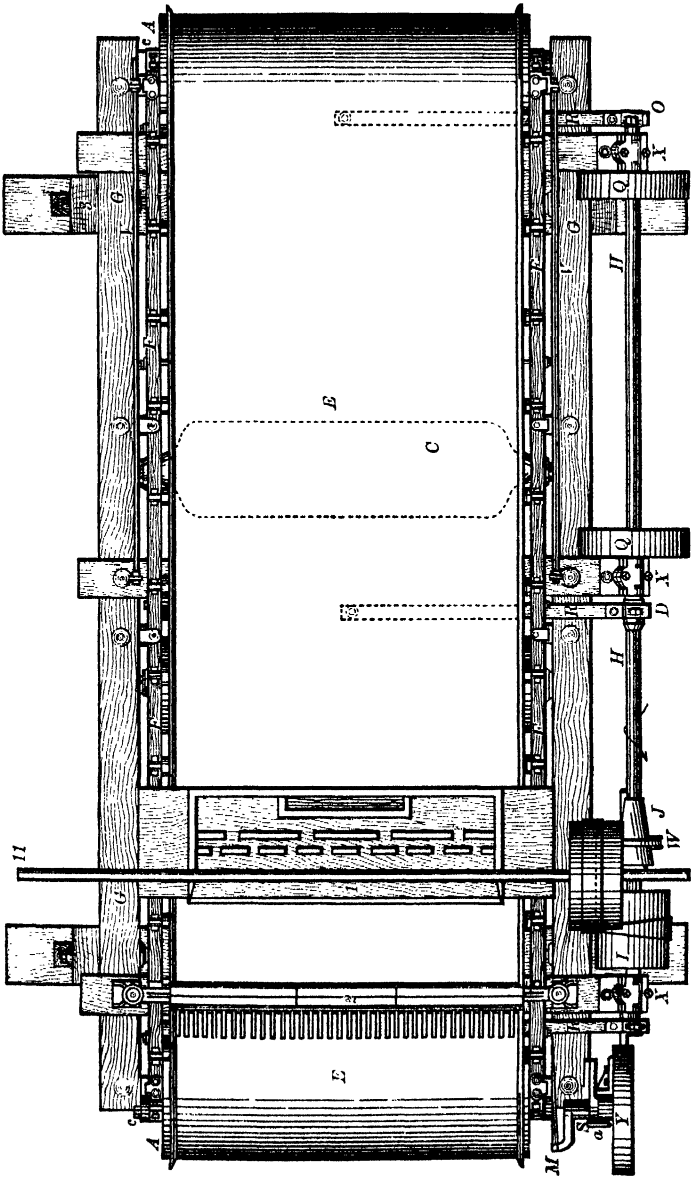


FIG. 75.—THE FRUE VANNER.

bearings of A, the upper or head roller, are higher than those of A, the foot roller; *i.e.* A is a trifle higher than the regular plane of the table, and the first small roller, D, should be raised a trifle.

The shape of the lower or bottom bearings of the uprights, N, can be understood by examining *b*, as shown in the end elevation and partly in the elevation. This lower bearing, *b*, extends across G, underneath, and is supported by a bolt passing through G. A lug on the upper side and on the outside end of *b* rests on G, and *b* hangs on the head of the bolt, and is kept stationary by the weight of *h* and its load. By striking with a hammer the face of *b* shown in the elevation, *b* is moved, changing the position of the lower bearing, and thus making N more or less vertical. By thus moving the lower supports of N, the sand corners in the belt, to be hereafter explained, are regulated.

The cross timbers binding together G G, and resting on them, are extended on one side, and on these extensions rests with its connections the main or crank shaft, H. This crank shaft has its bearings, X X X, and on them are brass cups to hold the lubricating compound; the cranks are ½ in. out of the centre, thus giving a one-inch throw. I is the driving pulley, forming with its belt the entire connection with the power. J is a cone pulley on the crank shaft, H. By shifting the small leather belt connecting J and W, the uphill travel of the main belt, E, is increased or diminished at will. The small belt connects to J the flanged pulley, W, which is on the small shaft, K, and by means of the hand wheel can be shifted on K and held in place.

The bearings of K are fastened to Y, a cast-iron shell protecting the worm, Q, and the worm gear, L; Y turns on a bearing bolted to the outside of G, and thus becomes a fulcrum for W and K. The object gained by this is that the weight of W and K (from Y) hangs on the small leather belt, preventing slipping or wear, at the same time making it positive. *a* is a screw used to relieve the small belt from the weight of K and W, taking all the strain off the small belt, and thus instantly stopping the uphill travel when desired. *m* is a hand screw by means of which the pulley can be moved, adjusting the small belt on the cone J, and thus regulating the uphill travel. K is the worm shaft, and terminates in a worm, Z, which connects with a worm gear, L. I travels in a bearing bolted to the outside of G. Z and L are protected from dirt by the shell of cast iron, Y, enveloping both (Fig. 76).

The short shaft which L revolves terminates in an arm, S, which drives a flat steel spring, M (which is a section of a circle), connected with the gudgeon of A. N N are the upright supports of the shaking table, F, carrying the belt, E. R are three flat steel spring connections bolted underneath the cross pieces of F, and attached to the cranks of the shaft, H, by brass

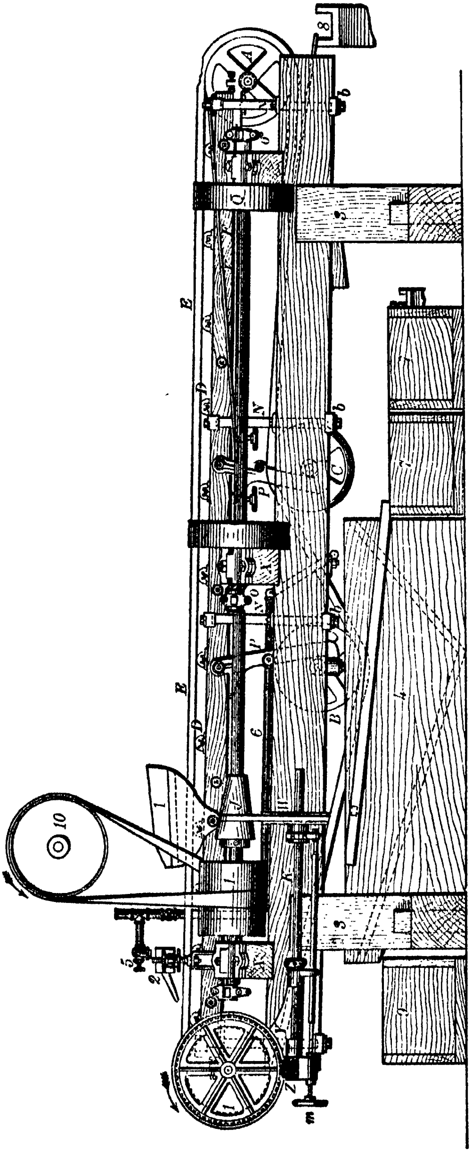


FIG. 76.—THE FRUE VANNER.

boxes, O, O, &c., on which are cups for the lubricating compounds. These springs give the quick lateral motion, about 200 a minute. Q Q are two fly-wheels. V V are two rods passing from the middle cross timber to the lugs for the same at the foot of F. The cast-iron washers on the bolts of the cross timbers have lugs cast on them. V V pass through these lugs, and at each end are nuts on each side of the lugs. Thus V V prevent the movable frame N from sliding either up or down, and by them F is squared.

In Fig. 77 No. 2 is the clear water distributor, and consists of a wooden trough supplied with water by a pipe. The water discharges on the belt in drops from grooves 3 in. apart. Another form generally used for No. 2 is that of an iron trough, having brass spouts 1½ in. apart; by blocking every other of these holes, water jets can be made 3 in. apart. No. 1 is the ore spreader, which moves with F, and delivers the ore and water evenly on the belt. N is a copper well that fits in (and shakes with) the ore spreader at the place shown in the figure. This is used in concentrating gold ores, for saving the amalgam and quicksilver which escape from the silvered plates above, and can be taken out and emptied at any time. Into this well falls all the pulp from the battery. Its ends are lower than the wooden blocks of the spreader, so that the pulp passes over the ends of the well and is evenly distributed. For some gold ores it is desirable to use on the ore-spreader a silvered copper-plate the size of the spreader, and when this is so the wooden blocks of the spreader are fastened to a movable frame on top, so that they can be removed when the plate is cleaned up once or twice a month.

Nos. 5, 5, are the cocks to regulate the water from the pipes, Nos. 6, 6. Nos. 3, 3, are upright posts, which are firmly fastened into two sills. These posts are cut down on the inside to make square shoulders, on which rest G G. No. 4 is the concentration box, in which the water is kept at the necessary height to wash the surface of the belt as it passes through it. The overflow from No. 4 contains finely divided sulphurets in suspension; to settle these the water passes through boxes, Nos. 7, 7, 7. No. 8 is a section of the launder which carries off the tailings. No. 9 is a box into which the concentrations fall when scraped out of No. 4. Nos. 10 and 11 show the arrangement of the countershaft with the tight and loose pulleys for driving the machine; but it should be higher above the machine than represented in the drawing (Fig. 77).

In working the Frue vanner, the ore is fed with water on the belt, E, by means of the spreader, No. 1. Thus the feed is spread uniformly across the belt. A small amount of clear water is distributed by No. 2, which is a wooden trough, and contains a pipe (No. 6), or by an iron trough with brass spouts.

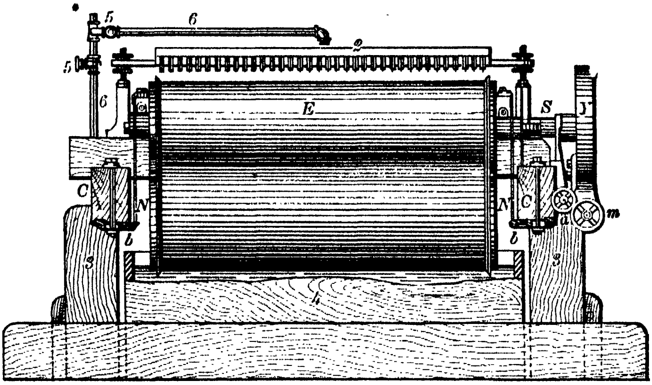


FIG. 77.—THE FRUE VANNER.

A depth of $\frac{3}{8}$ to $\frac{1}{2}$ in. of sand and water is constantly kept on the table. To the main shaft, H, sufficient motion is given to impart from 180 to 200 revolutions, with 1 in. throw.

The uphill travel or progressive motion varies from 2 to 12 ft. a minute according to the ore, and the inclination of the table is from 3 to 6 ins. in 12 ft., varying with the ore. This inclination can be changed at will by wedges at the foot of the machine; these wedges are under the lower end of G G, and rest on the shoulders of the uprights from the main timber of the mill.

The motion, the water used, the inclination of the table, and the uphill travel, will have to be regulated for each description of ore, but this once done no further trouble should be experienced in the manipulation.

In treating ore from the stamps, it may be found that too much water has been used by the stamps for proper treatment of the sand by the machine: in such a case there should be a box between the stamps and concentrator, from the bottom of which the sand, with the proper amount of water, may be drawn, the superfluous water passing away at the top; but as mineral matter will also pass away with it, there should be settling tanks to receive the water; the settlings can then be worked from time to time as they accumulate.

Proper Consistency of Pulp in the Frue Vanner.—The use of a proper quantity of water with the pulp from the stamps, and also its proper regulation, are very important. There should be formed on each side of the belt a slight corner of sand—that is to say, there should be on each side sand with less water in it than there is in the balance of the pulp on the belt. Unless this is so the corners will be sloppy, and there will be a loss. Sloppy corners are caused by using too much water with the pulp from the stamps passing on No. 1.

On the other hand, there may not be enough water with the pulp from the stamps, and the result will be too heavy sand corners. The remedy for this is to use more water in the pulp coming on No. 1.

As regards the proper amount of water to be used in the water spreader, No. 2, just enough and no more should be used to keep the ground between No. 1 and No. 2 covered, so that no points or fingers of sand shall show on the surface. The whole width of the belt between the water spreader and the ore spreader should be kept quite wet. If dry streaks or points occur, and the water as a consequence runs in streaks at the junction of the wet and dry channels, mineral will be picked up and floated away on the surface of the water, which "floating" of mineral is caused by its dryness, and not by its lightness; it has been coated with a film of air.

The proper amount of water with the pulp on No. 1 and the proper amount of water in No. 2 being fixed, the carrying over of the clean concentrations past the jets of No. 2 should be accomplished and regulated by the uphill travel only.

Frequently the sand and water on the belt will be distributed unevenly, the sand working to one side of the belt and making a heavy broad corner, while the other is sloppy. To control and remedy this, see first that there is no jar about the machine; that there are no loosely working parts; that everything is working noiselessly; and that all the parts are in line. If there is not an exact balance of the pulp on the belt, the heavy sand corner forms on one side or the other. To adjust the load and keep the sands evenly distributed on the belt, the lower bearings, *b*, of all the uprights, N, on one side of the machine, are moved forwards or backwards by slight blows of the hammer. The change of position from the vertical of N, &c., thus occasioned, affects the pulp on the belt; and by changing the position of *b*, &c., on one side or the other, the right balance or equilibrium will be obtained, and the sand and water (or pulp) will be uniformly distributed across the belt. If the heavy sand corner is on the shaft side, the bottom bearings, *b*, &c., on the opposite side must be moved out.

Again, the sand corner can be partly controlled by bending the end of the driving spring, fastened in the collar, towards the side having the thickest sands. The same effect, and even more positive, is produced by moving the crank shaft, and with it the table, the same way as the end of the driving spring is bent. The rolls underneath have also an effect on the corners, by swinging one end of each either towards one another or in the opposite direction.

The water in the concentration box is constantly agitated by the motion of the belt, and consequently the water escaping from this box carries in suspension a good amount of very finely divided sulphurets of high assay value. To save these, settling boxes, Nos. 7, 7, 7, should be used, which can be cleaned out once a month, a product being obtained which will add materially to the value obtained from the ore.

The quantity of water required for one machine will be from 1 to 1½ gals. per minute of clear water at the head, and from 1½ to 3 gals. per minute with the pulp. The boiler for a 5-stamp mill, with two concentrators, requires 1 gal. a minute; hence, in places where water is very scarce, 2 gals. per minute can supply five stamps, two Frue ore concentrators, and the boiler, by settling and pumping back.

The Operation of the Frue Vanner.—Recent practice has demonstrated that about 6 tons per twenty-four hours, passing through a 40-mesh screen, is as much as it is advisable to treat in a single machine. If a battery of five stamps does its proper amount of work, the quantity crushed is largely in excess of 6 tons; and for this reason the best practice is to have two Frue vanners for five stamps, if the stamps are heavy and the sulphurets are of high grade and difficult to save. Where pulp from five stamps is fed to two machines the pulp is divided, one half passing on each. The machines are generally placed in a double row on the same level, head to head, so that the attendant overlooks both rows in walking between. The concentrator floor should be so far below the level of the battery as to allow the feed launder to be above the head of the attendant. No sizing of the material is needed; the pulp passes directly from the stamps on to the copper plates (if used), and thence on to the vanners.

The belt forms the bed on which the dressing of the ore is effected, being an inclined plane 12 ft. long, and having down the two sides projecting rubber flanges, which prevent the water and sand from dropping over the sides. The arrangement of rollers permits of the belt being slowly revolved in the direction of its length and *up* the incline; thus, though the direction of the working plane remains always the same, its surface is constantly travelling. The crushed rock in a small stream of water falls near the upper end of the belt, by means of the sand distributor, No. 1, and flows down the belt towards its lower end. Now, as the inclination at which the belt is set is only from three to six inches in twelve feet, and as the stream of water is not large, and spreads also over the whole width of four feet, it is obvious that much of the crushed rock contained in the water would settle on the belt, while the water and the finer and lighter particles of sand would alone reach the foot of the table and drop over into a waste launder. In addition, we have the travelling of the belt upwards and onwards continuously.

The effect of this would naturally be to deliver all the rock which has settled on the belt over the upper end roller, *a*, and deposit the same eventually in the water tank, No. 4, below, through which the belt passes in plying around the roller, *b*. The action of the belt, then, simply amounts to this, that it forms an inclined plane, or working surface, which, by its progressive motion, will deposit in the tank, No. 4, all solid material which settles on it—that is, whatever is not carried off in suspension by the water flowing from No. 1. The belt is merely a self-discharging bed. To separate the heavier metallic minerals from the accompanying gangue or rock, it is evident that the above described action of the belt is not sufficient, for not only would the mineral be delivered in the tank below, but also a large proportion of the rock, which would certainly settle on the belt as well. A separation of the two classes has yet to be accomplished. For this purpose a second stream of water is employed. About one foot above the sand distributor—that is, just below the first small roller, D, —the water distributor is arranged, which delivers small jets of water, three inches apart, over the entire width of the belt. The revolving belt, carrying its load of settled rock and mineral, travels past the jets of water, taking with it such particles of mineral as have weight or specific gravity sufficient to resist the force of the descending water, while the lighter particles of rock are driven back by the water, and do not reach the tank, No. 4. In addition, a gentle side shake is given to the belt at right angles to the length and travel of the belt. By the introduction of this secondary motion the sand is kept in gentle agitation, uniformly distributed over the whole width of the belt, and the heavier particles of mineral, settling through the sand, cling to the belt and are carried up it, past the small jets of water, and deposited in a cleaned state within the tank for collection. Very little water is now needed to effect the separation of rock from mineral.

This machine is not adapted for the treatment of very coarse material, such as would be submitted to "jigging," but is adapted for the treatment of fine sands and slimes. Many forms of washing apparatus have been introduced for the special treatment of slimes. Some of these have been already described, such as the buddle, the inclined table, and the blanket sluice.

The important feature of the Frue vanner lies in the property which fine ore particles have of clinging to the rubber belt. The shaking motion separates the mineral from the sand as it flows slowly down the belt, as it keeps the material in gentle motion, and when once the material has touched the surface of the belt it clings and is carried up past the small streams of water at the head of the machine, and is dropped as the belt passes, in a reversed position, through the water tank.

The side shake communicated to the belt is of the utmost advantage in more ways than by the settling of the mineral from the sand; for by keeping all the material in motion the belt can be set at a slighter angle, a smaller quantity of water used, and a much greater quantity of material operated on, than would be the case if a simple belt, without lateral movement, were employed. The sand does not pack and cause the water to cut channels and to run off in small streams, but is always uniformly distributed over the whole width of the belt.

As regards the ores on which the machine will work, the only point of importance is, that there be a fair difference between the specific gravity of the mineral to be saved and that of the waste matter with it. The following minerals have been worked upon with excellent results: iron and copper pyrites, arsenical iron pyrites, zinc blend, galena, tinstone, cinnabar, native silver, carbonates of lead and copper and native copper, tellurides of gold and silver, and tailings from the amalgamating mills. "Floured" quicksilver and slimes flowing from the settling tanks have also been experimented on, and made to yield the impalpable mineral which they contain.

For running a single machine it is estimated that only one-quarter horse-power is required, and one man can attend to sixteen machines without difficulty. When six machines are used the cost of treated sands, when ready to flow on to the machine, is estimated to be about tenpence per ton.

The revolution of the belt is the agency by which the delivery of the clean material is effected. The necessity for a proper travel will be apparent, inasmuch as, supposing the belt to remain stationary, no delivery of mineral could possibly take place; while, if a quick travel be communicated, everything which fell on the belt from the sand distributor, No. 1, would be rushed past the clear water at No. 2, and collected in the tank. Between these extremes we find the desired mean: a speed which shall be sufficiently great to deliver continuously all the mineral collected by the belt, yet not so fast as to require a flood of water at No. 2 to keep back the sands. If the ore treated be poor in mineral, the upward motion of the belt should not exceed twenty inches per minute; if rich, the speed is increased accordingly, and in agreement with the inclination of the belt, being greater as the inclination increases; but usually the speed should not exceed 3½ ft. per minute.

To examine the influence of the side shake two extreme cases may be cited. In the absence of side movement, with the ordinary supply of material coming on to the belt, no separation can be effected by a reasonable stream of water at No. 2; the greater part of the rock passes over into a tank with the mineral it packs upon the belt. On the other hand, to drive the crank-shaft, H, at a rapid rate, and thus to violently agitate the belt and its load, has the effect of working everything off the foot of the table. In this matter, as with the revolution of the belt, there is clearly a desirable mean; namely, a speed at which the material on the belt is kept in gentle motion —lightly suspended in the water, and thus easily carried by it down the belt—a speed which allows and facilitates the settlement of the mineral from the rock, and so does not disturb it when once settled on the belt. The customary rate of driving this side motion varies from one hundred and eighty to two hundred revolutions of the shaft per minute—the former speed for fine, light slimes, and the latter for rough and heavy sands.

As regards the regulation of the water delivered at No. 1, the ground between No. 1 and No. 2 should be kept nicely covered with water, and the mineral brought through by regulating the uphill travel. To make the final separation of mineral from sand some little judgment is necessary. As already stated, the delivery holes in the water launder are 1½ in. apart across the whole width of the belt, and the clear mineral creeps up between these small jets of water, so that, as delivered over the head of the belt at A, the form is that of longitudinal streaks, farther or nearer apart, and of greater or less width according as the richness of the material treated is different. The primary object in the adjustment of the uphill travel is that the clean mineral shall be allowed to pass over into the tank at the same rate as it is fed on to the belt in the mixture to be separated. For example, suppose that in every hour 800 lbs. of mixed rock and mineral pass on to the belt, and that the mixture contains 5 per cent. of heavy mineral, say galena. Now, disregarding the small loss of mineral in the proper waste or tailings, the uphill travel must be so regulated that there shall be a steady delivery of mineral at the rate of 40 lbs. per hour. No more than this can possibly be delivered unless rocky impurities are allowed to pass and to be weighed in; and if less than this passes, there must be a continual accumulation of mineral on the belt, which will eventually produce loss in the waste. The proper adjustment of the uphill travel may at first, perhaps, appear difficult, but in reality it is very simple, in which the eye furnishes a sure guide. The gauge by which the adjustment is rendered easy is the extent of head of mineral showing at the point No. 2, where the water strikes the belt. Again, the weight of mineral as it gets strong and heavy forces it more past the water. Should the discharge of mineral exceed the quantity falling on the belt, sand or rock will be found close up to the jets of water, and by-and-by passing them in place of mineral. If the uphill travel be too slow, the mineral collects below No. 2, forming a great head extending towards No. 1, and even below, in which latter case an increased loss of mineral will assuredly take place in the waste.

When working properly, a small head is always kept below the jets of clear water, and the mineral comes over clean and regularly. A few hours' experience will instruct any one sufficiently on this fact; and having once adjusted the uphill travel, the machine will work continuously and uniformly as long as the conditions are kept constant; nothing more than this can be expected of any machine.

The machine should work almost noiselessly; if there be any jar or knock, the cause must be found and a remedy applied. These jars can be easily remedied, and are not faults in the machine, but of its setting up or adjustment. If the side shake be found to work smoothly and without a jar, the uphill travel or progressive motion can be given. The machine is now at work, and some clear water run on it from No. 2 will show whether the belt is level across; if not, it is easily levelled by the wedges at the foot before the ore feed is started.

Supposing all instructions to be followed, the machine will be working now regularly and smoothly. The belt, moving always onwards, brings all mineral up to the clear water at No. 2; and here the difference between rock and mineral becomes apparent: the clean mineral passes between the jets of water, and is deposited in the tank below; the sand works gradually down, to be replaced by other particles.

In treating slimes, as indeed with all other qualities of material, as little water as practicable should be fed on with it; a large volume of water on an inclined plane surface implies speed and force—two undesirable elements in the separation of fine mineral. From an extended experience with the machine it has been found that with a slightly increased speed of the side motion, any rough particles of rock are much more easily moved than fine mineral; that it is easy to work the coarse sand off the belt, and at the same time produce extremely slight loss even of the very finest mineral. This observation led to the working of mixtures of sizes which should properly, on the usually accepted theory, have been classified and treated separately. It is preferable to use a screen of forty holes to the lineal inch. The side motion of the belt works off the rock and never moves the very finest mineral when it has once touched the belt. Therefore the pulp flowing on the belt must not be too thick, as the particles of mineral cannot settle through it. For this reason a pretty fair current of water must be allowed to go on with the slimes, and the belt placed with very slight inclination, so that the current be not too rapid.

Concentration of the Sulphurets.—The success of concentration is dependent on the efficiency of the machinery employed to effect a separation of the worthless gangue from the valuable portions of the ores, and it is very difficult to lay down any rule for the guide of the mine owner in the choice of machinery for the purpose. Almost every mine having its own distinct character in regard to ores, no one machine can be adapted to the varying physical characteristics of the numerous classes of ores. But as a rule, in most ores, the gangue is harder than the mineralized portion, and the result in stamping is that the valuable ore particles are generally reduced much finer than is consistent with economical dressing. Moreover, even pulverization—a condition which is essential to perfect concentration—is difficult to obtain in practice. If ores are of a complex nature, and the precious metal is contained in minerals of varying densities and different degrees of hardness, an even pulverization becomes still more difficult.

The best method, therefore, is to effect an equal assorting or sizing of the uneven grains resulting from the stamping process before proceeding to concentration.

Colonel Taylor on Concentration.—A practical illustration of the difficulties met with in concentration has been given by an experienced Californian, Colonel J. M. Taylor, who gives the results of his observations and experiments as follows:—

"I commenced this business believing, as many other theorists have done, that all gold ores could be concentrated on coming from the battery by machinery without handling, and that gold sulphurets could be treated successfully only by the chlorine process. I continued in this belief for many years, spending time and money, and accomplishing nothing. I purchased a mine, for experimental purposes, which had the reputation of producing ore of a very refractory character. This ore assayed $30 per ton, but not more than one fourth of it was sufficiently free to admit of its being amalgamated in the battery.

"I erected a five-stamp mill, and tried various methods for concentrating the sulphurets. The best result obtained was 25 per cent., which, together with the free gold, formed only 50 per cent. of the assay value of the ore. At this juncture I abandoned everything with the word patent on it, and, going back to first principles, constructed an old-fashioned Cornish buddle, and sized the ore in two sizes, using two pointed boxes, after the plan adopted by the most improved mills in Grass Valley; all the materials held in suspension by the water were allowed to pass over the second box and go to waste. I found by concentrating the two sizes separately in the buddle that I could get about 8 per cent. more than when they were concentrated together. In this way 10 per cent. more was saved than by any other plan yet tried.

"The tailings as they came from the buddle were assayed, and found to contain 10 per cent., leaving 30 per cent. unaccounted for. A tank was then constructed 12 by 12 ft., with a partition in the centre, and the slum that ran over the second box was allowed to pass into the one and out of the other, giving it plenty of time to settle. In this way one fifth of all the ore crushed was settled in the tanks, the contents of which assayed 23 per cent., being at the rate of about $6 per ton of ore, making a saving of an assay of $48 per day with an eight-ton mill. Deducting from this 10 per cent. for loss in concentrating, 25 per cent. for working, and \$1 per ton for cost of concentrating, resulted in a net profit of \$25 per day, or \$720 per month, to the mill. A barrel holding 60 gallons was placed under the stream of water from the tank, and when full was left twenty-four hours to settle, a little alum having been added. The top was then carefully poured off, when the sediment was found to contain about 1 per cent. of the ore, which was held in suspension by the water after it had become comparatively clear; 8 per cent. could not be accounted for. It could easily have been wasted in the battery, or more than an average might have been got in sampling the mine. This latter was hardly possible, however, as great care was taken to insure a fair sampling by drilling through the ledge in various places.

"This result did not surprise me in the least, having long been aware that a large percentage from most mills had been lost in this way. The question was how to concentrate these tailings up to a higher grade, they not being rich enough to pay for chlorinating. Various methods for accomplishing this were tried. The best result from the round buddle, using an ordinary broom for sweeping, was 50 per cent. A buddle was then constructed on a larger scale, and with much less grade than the one already in use. A piece of common mill blanket was put on the arm for sweeping, and a small stream of water turned on. This proved a success as shown by assay, twelve tons having been reduced to one ton, at a cost of 75 cents per ton. There was still a loss of ten per cent. Various tests were made in order to determine what grade of sulphuret ore would pay to concentrate. Some twenty tons of coarse tailings had accumulated from the buddle, which after testing proved to be worth \$2.25 per ton. This was reduced, at a cost of \$7½, to 500 lbs., which had an assay value of \$30. Deducting from this 25 per cent. for loss in working, and \$7½ for labour of concentrating, left a net profit of \$14.

"I now became satisfied that no machine yet invented can concentrate the majority of ores to more than fifty per cent. of their assay value without their having first been sized and settled in tanks. Assuming that ore requires settling before it can be concentrated up to a high percentage, it is only a waste of time and money to attempt its concentration before settling, as the cost is the same whether it be high or low grade. As a consequence, any machine that fails to take out more than half the value of the ore is of no practical use. All ores must be sized in three different sizes before they can be concentrated. Lead sulphurets are nine and one half times heavier than water, and five times heavier than quartz. Common iron or copper sulphurets are seven times heavier than water, and three times heavier than quartz. The coarse pulp and sulphurets capable of passing through an ordinary No. 6 mill screen are, perhaps, on an average, fifty times coarser than those found in the slum ores. They should, therefore, be concentrated separately; otherwise a current of water sufficient to carry off the coarse pulp will also carry off the sulphurets, notwithstanding the latter are from three to five times heavier than the pulp, which, being composed of quartz about fifty times more bulky than the sulphurets, exposes a corresponding surface to the action of the water.

"To obviate this trouble, we must equalise the tailings, bringing the sulphurets and the pulp to the same size, then equalise the water to correspond with the fineness of the pulp, and a current that will carry off the latter will leave the heavier sulphurets behind. No man experienced in milling will ever spend a dollar trying to concentrate in violation of these rules or natural laws. Any of the quick-motioned concentrators now in use will separate a large percentage of the coarser sulphurets from the coarse sand, but at the same time they will hold the fine sulphurets, which are of the most value, in suspension so long as the water continues in motion. The round convex buddle is the best equaliser in use, it being fed around the centre-post, which is about twelve inches in diameter. As the water recedes from the centre it spreads, and consequently decreases in force. If the current of water be strong enough to start the finest sulphurets from the head of the buddle, it will become so diminished before reaching half the distance from the centre to the circumference that the sulphurets are left behind. The concave or centre-discharging buddle is fed on the outer rim and discharged in the centre: consequently the water increases in force toward the centre; hence fine sulphurets leaving the circumference of the buddle will be carried toward the centre with the pulp by the increasing force of water.

"In the tin, lead, and copper mines of England, where concentration has been carried to a higher state of perfection than anywhere else, they have long since discarded the centre-discharging buddle, and use only the convex. Many mining superintendents contend that their ores are not rich enough to justify handling and concentrating in round buddles, but I am of opinion that all ores below permanent water level will pay to concentrate if they will pay to work at all.

"After the concentration was perfected, two hundred tons of ore were run through the mill, and further concentrated to fifteen tons, which were shown by assay to contain ninety-one per cent. of the gold found in the ore after being settled in the tanks and before it was concentrated. This second concentration costs 63½ cents per ton."

CHAPTER VI.

TREATMENT OF GOLD-BEARING ORES: CHLORINATION.

The Plattner Process of Chlorination according to Prof. Küstel—Assay of Gold Sulphurets—Chlorination Process for Sulphurets and Arseniurets—Roasting Furnaces—Pontgibaud Furnace—Küstel's Furnace—Bruckner's Cylinder—Treatment of the Ore with Chlorine Gas—Cost of Treatment by Plattner's Method—Loss of Gold in Chloridising Roasting—Proposed Condensation of Fumes by Electricity—The Plattner Process at Plymouth Mine.

The Process of Chlorination.—The Plattner process for the extraction of gold by means of chlorination, as improved by Mr. Deetkin, of Grass Valley (an account whereof, to which I am indebted, has been given by the late Professor Küstel *), may be described in general terms as follows :—

(1.) The auriferous concentrates from the stamping mill having been perfectly oxidized, are moistened with water and put lightly, by means of a sieve, into a wooden vat, coated with tar and rosin, and having a perforated false bottom upon which a filter is made, for which there are numerous ways. When filled, a close-fitting cover is placed on top.

(2.) Chlorine gas, produced by decomposing salt and peroxide of manganese with sulphuric acid, is introduced between the false and true bottoms, and made to permeate upwards through the ore mass. After the expiration of from fifteen to forty-eight hours, the gas is found to appear abundantly on the ore mass, and is then shut off, and the vat allowed to remain a few hours under the influence of the gas. The cover being removed, pure water is added to fill the vat even with the top surface of the ore; the fine particles of gold, under the action of chlorine, have changed from metal to a soluble terchloride, and in this condition it is drawn off or leached out with water, fresh water being added until a test shows no trace of gold.

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* In his "Treatise on Concentration of all kinds of Ores, including the Chlorination Process." San Francisco, 1868.

(3.) A prepared solution of sulphate of iron—the usual precipitant—is carefully added to this drawn-off solution, and the gold thrown down as a black or brownish precipitate; this is gathered, washed, and melted into ingots of nearly pure gold.

Conditions for Successful Chlorination.—The process is thus based upon the property of chlorine gas to transform metallic gold into soluble chloride of gold, and with some kinds of pyrites it is very perfect if well executed, but the following requirements have to be carefully observed:—

(1.) The gold must always be in a metallic state. Quartz, free from other earths and sulphurets, containing very fine gold, can be subjected to chlorination without other preparation than moistening with water, as described farther on. Sulphuretted ore requires a perfect roasting. The presence of lead makes a careful roasting necessary, commencing with a very low temperature. All metals, except gold, must be transformed into oxides. Sulphates are injurious.

(2.) The chlorine gas must be free from muriatic acid. From the generator the gas is forced through clear water, by which the muriatic acid is absorbed. The muriatic acid dissolves the oxides, and causes, when sulphides are present in consequence of defective roasting, the formation of sulphuretted hydrogen, by which soluble chloride of gold is precipitated. The muriatic acid dissolves also oxides of metals precipitated by the addition of sulphate of iron with the gold.

(3.) There must be no other substances in the charge which will unite with the free chlorine, since this would occasion a great waste of gas, and a failure in the desired separation of gold from other metals.

(4.) There must be no reaction in the mass treated with chlorine which will prematurely precipitate the gold before the final solution is obtained and drawn off.

(5.) In a word, it is required that all the gold, and, if possible, nothing else, shall be obtained in the final solution. Precipitation and melting then present no special difficulties.

Generally the concentrated sulphurets from the gold mills are subjected to the chlorination process; but also ores consisting of quartz and free gold, without admixture of other earths or sulphurets, can be treated by this process without any further preliminary treatment than reduction to powder.

Assay of Gold Sulphurets by Chlorination.—Of the finely pulverised sulphurets, from five to seven ounces or more are weighed out and roasted on a piece of sheet iron, the edges of which are bent up, and the inside coated over several times with clay water and then well dried.

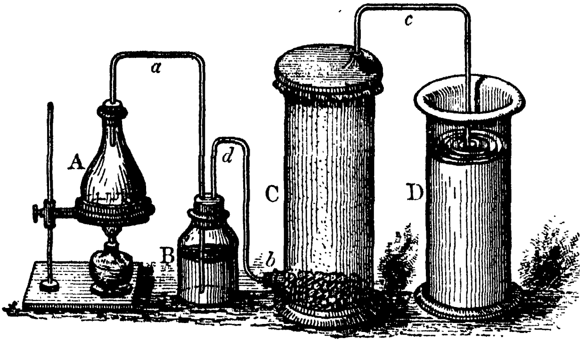

Fig. 78.—Chlorination Test Apparatus.

The roasting may be performed over charcoal or coke, in a small stove, or in a large black-lead crucible, through the bottom of which a hole is cut for the draft. The sulphurets must be stirred with an iron spatula until no sulphurous smell is perceptible; after which a tolerably strong red heat is applied. When cold, the sample must be ground over in an iron mortar and roasted once more at a red heat. When no sulphurous smell is observable the roasting may be considered as finished.

When cold, the roasted ore must be moistened with sufficient water in a dish or cup to make it of a loose or woolly consistence, in which state it is best suited for chlorination. If the roasting be perfect, the metallic gold can be dissolved in chlorinated water and extracted; but in case there be in the roasted stuff a small quantity of sulphurets or arseniurets not decomposed, it is more proper to use chlorine gas. The extraction is performed in the following way:—

Into a glass cylinder, C (Fig. 78), from 8 to 10 in. high, and 2½ in. wide, provided with a neck, *b*, near the bottom and about ¾ in. wide, is introduced a layer of small clean quartz fragments as shown in the drawing, and on this a thin layer of coarse and then of fine quartz sand. This quartz forms the filter; pulverised glass can also be employed. Over the quartz is then placed the roasted and moistened ore, as loosely as possible. The cylinder has a cover of wood or of india-rubber, in which a glass pipe is fixed, as shown at *c*. The longer end of this pipe dips into another cylinder, containing rolled stiff blotting-paper or pieces of blotting-paper, or shavings of wood, moistened with alcohol.

For the generation of the chlorine—say for 20 ozs. of roasted sulphurets—a glass vessel, A, is charged with 1 oz. of pulverised peroxide of manganese, 4 ozs. of muriatic acid, and 1 oz. of sulphuric acid, mixed with 1 oz. of water.

This mixture is shaken up, and the vessel placed on a cup or on a piece of sheet-iron covered with sand, with the neck corked. Through the cork a glass tube, *a*, is fixed so that it shall reach about 2 in. below the surface of the water contained in the bottle, B.

The chlorine gas is forced through the water, and by this means washed from muriatic acid. It is then conveyed through the pipe, *d*, into the ore receiver, C. The apparatus is more conveniently arranged if the three tubes, *a*, *d*, *c*, are each made in two parts, the first and last joining at *a* and *c*, by short india-rubber tubes. The tube, *d*, is connected below, near the cork, *b*, so that its short horizontal part will serve for the discharge of the lixivial when disconnected. As the chlorine is not only disagreeable but also injurious, it is necessary to cover all corks and joints with wheat flour dough. The above quantity of sulphurets (20 ozs.) requires about two hours' roasting, and a receiver, C, 6 in. high and 4 in. diameter.

The operation can be performed in a room without unpleasantness from the chlorine, because, with the alcohol in the cylinder, D, it forms chloral and muriatic acid, which are not disagreeable.

The mixture in the vessel, A, must be moderately heated at the beginning. The gas appears of a greenish colour, and soon fills the cylinder, C, by which time the finest gold is transformed into chloride; but this is not the case with the coarser particles, and with such gold as may be contained in sulphurets not decomposed. For this reason the development of the gas must continue for at least one hour more, before the chlorination is finished.

If auriferous pyrites is operated upon, the apparatus can be taken apart, commencing always with the tube, *a*, otherwise the water from B would be drawn over to A. If compound ores (galena, zinc, &c.), with free gold, are under treatment, the bottle, A, is separated from the apparatus, which remains undisturbed for at least fifteen hours. After this time, when all is taken apart, warm water is carefully introduced over the ore, in order to lixiviate the chloride of gold and other soluble salts. The cylinder must be a little inclined, so that no fluid shall remain on the bottom, and if the neck should be a little too high above the bottom, this can be filled level with the outlet with pitch.

The solution obtained must be mixed first with a few drops of muriatic acid; then a clear solution of sulphate of iron (green vitriol), in sufficient quantity, added, and stirred with a glass rod. The whole is allowed to stand (if possible, warm) till all the gold is precipitated and the solution clear. If a few drops of the sulphate solution should effect a slight precipitate when added to the clear fluid of the precipitated gold, it would prove that too little of the precipitant was used. The solution and precipitated gold are introduced into a filter, and washed with clean water: then the gold is dried with the filter, and washed with clean water; after which the gold is again dried with the filter in a porcelain cup or in an assayer's dry cup, and then burned, with free access of air. When cold, the ash is mixed with one or two hundred grains of test lead and cupelled under the muffle, and the gold button weighed. If gold quartz containing only fine gold and no sulphurets is under treatment, roasting is not required; but it is necessary to extract the metallic iron—which results from pulverising in an iron mortar—by means of a magnet. In place of the cylinder, c, if such cannot be obtained, a bottle, the bottom of which is cut off by means of a string, as shown in Fig. 79, will answer the purpose. Or a wooden box of a square section may be substituted for the cylinder, provided the inside be carefully coated with pitch or asphaltum cement.

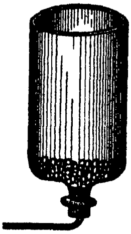
FIG. 79. CHLORINATION VESSEL.

Chlorination Process for Sulphurets and Arseniurets.—From what has been said before, it will be seen that it is essential that each gold mill should be provided with proper concentrating apparatus, by which the worthless gangue may be separated from the heavier and valuable ore particles. Gold ores generally carry a certain percentage of sulphurets and arseniurets which are auriferous, and these are collected on the concentrators along with some fine gold which has escaped the battery and copper-plate amalgamation. If any silver is in the ore, the mineral particles are also collected on the concentrators; but by this it is not implied that concentration in every case will prove so perfect as to collect all that is valuable in the ore, and a careful watch should be kept over the tailings which leave the mill. Frequent assays of the tailings should be made, so as to guide the operator in his manipulations.

Besides iron pyrites and arsenical pyrites, the ores often contain copper pyrites, galena, zinc blende, tellurides, &c., and whatever is collected on the shaking tables or runners is designated by the generic name of *concentrates*. The nature of the ore particles constituting the concentrates will determine the results of the chlorination process and percentage of precious metal extracted from them.

It is a mistaken idea which induces certain operators to allow the concentrates to lie for any length of time exposed to atmospheric action, whereby a natural decomposition is expected to free some of the precious metals, and where nature is to effect a similar change to what would take place in the roasting furnace. To a certain extent this may be true: the concentrates do oxidize, but they clog together and form hard lumps which have to be re-pulverized, and it is therefore better to dry them at once and then to roast them.

There are not many establishments in existence where the ores are pulverized and treated at once by the Plattner process, without concentration, but since the introduction of the Mears process, which will be described presently, it has been found practicable to carry out the system with certain classes of free gold ores; and no doubt before long, with an economical production of chlorine, proper appliances, and rapidity of working, the amalgamation process will have to give way to chlorination whenever the gangue will permit of such treatment.

Location of Chlorination Works.—This must be chosen with reference to the prevailing winds, the fall of the ground, and the ready supply of clean water. As the sulphuric acid and chlorine vapours are destructive to the machinery of the mine and mill, great care is necessary to select a site for the furnace from which they cannot be carried to the mill by the prevailing winds. The direction of the draught is also in that of the winds. For the proper and economical working of the process a fall of at least thirty to forty feet is required. The water supply, of at least thirty-five gallons per hour, is discharged into a wooden tank which will hold a twenty-four hours' run.

The concentrates are delivered at the ore floor of the chlorination works, on a level with the charging hopper of the upper furnace. They contain on an average six per cent. of moisture.

Roasting of the Sulphurets.—The presence of *galena* and of *copper sulphurets* does not interfere with the chlorination, provided the roasting be well performed. Gold of very low fineness, containing from forty to fifty per cent. of silver, will probably resist the chlorination, unless it is in the finest state of pulverisation, as the silver forms an impenetrable coating of chloride of silver, which can only be removed by a chlorinated solution of salt, which dissolves the chloride of silver and also that of gold. The formation of chloride of silver would prevent the thorough chlorination of the gold particles. The silver combined with the gold is not obtained by the chlorination process, but only the pure gold, for the reason that the amount of silver is generally too insignificant to admit of a remunerative extraction by a hot solution of salt. Coarse gold requires too much time to be converted into a chloride, so that the ore containing such gold is unfit for chlorination. Pan tailings of concentrated sulphurets allow an easy and perfect extraction of gold by this process.

The presence of *lime and talc* (*silicate of magnesia*) makes the chlorination of roasted ore very troublesome, or even impossible, but the addition of salt in roasting removes this difficulty. If the ore contains lead, the lead may be precipitated by sulphuric acid ; it is then necessary to allow at least two hours for the precipitation. This is especially required if there is a great deal of lead in the ore and the roasting was performed with salt. After the precipitate is deposited on the bottom, the clear solution containing the gold must be drawn over into another precipitating vat, and the gold precipitated by sulphate of iron ; or the gold is first precipitated by sulphuretted hydrogen, by which the lime, baryta, &c., which would be precipitated as a white powder by the sulphate of iron, remain dissolved in the solution.

The concentrated sulphurets must be subjected to roasting while damp, or at least before crusts and lumps are formed by oxidation, in consequence of lying too long. Pan tailings of sulphurets, however, must be dried, either artificially or by exposure to air and sun, and then pulverised by some means so fine as to admit of being sifted through a sieve of from twelve to fourteen meshes to the running inch. If these tailings were subjected directly to roasting they would bake into hard lumps, and the mass would be rendered unfit for chlorination. In all furnaces, at the beginning of roasting, at a low heat, the sulphur of the sulphurets is set free, and combining with the oxygen of the air produces volatile sulphurous acid, which is well known by its sharp, suffocating odour. The metals, by losing a part of their sulphur, are converted into oxides and sulphates, in which the iron sulphates are predominant. Sulphate of iron is the precipitant of the chloride of gold; its presence in the roasted ore is therefore very objectionable, and it is necessary to increase the heat by degrees, in order to decompose the sulphates and to form oxides.

Roasting Arsenical Pyrites.—Similar to this is the behaviour of auriferous arsenical pyrites when subjected to roasting. Arsenious acid escapes under the influence of heat and oxygen, while oxides and arseniates remain, the latter being further decomposed to oxides by increased heat. At the same time all metallic iron derived from grinding or stamping must be converted into an oxide.

After the sulphur and arsenic have been expelled, the gold remains in a free, metallic condition, and can be easily detected by pulverising and washing a small portion of the roasted stuff. While the formation of oxides and sulphates or arseniates is going on the gold is set free, and remains so during the whole process. When salt is used in roasting, the gold, according to Plattner, first forms chloride of gold ($Au\ Cl^3$) far below red heat. At 200° C. it loses a part of its chlorine, and at about 240° C. changes into a sub-chloride ($Au\ Cl$), which is not soluble in cold water. At a red heat it is converted into metallic gold.

There is hardly any loss of gold in roasting, and only in cases where the roasting is performed very rapidly are the finest gold particles carried off by the volatile products of roasting. In submitting, however, auriferous silver ores to a chloridising roasting with salt, great care has to be used, as during such roasting large portions of gold may become volatilized, a experiments in Australia and California have shown. Experiments made in the latter country have also shown that sulphurets which have been roasted for more than thirty hour without interruption have yielded 90 per cent. of the gold a compared with the fire assay. The tailings always containe some gold, so that very little or none of it was lost during th roasting.

The Roasting Furnaces.—*Single Roasting Furnaces.*—Figs. 80 and 81 show the longitudinal section and plan of a

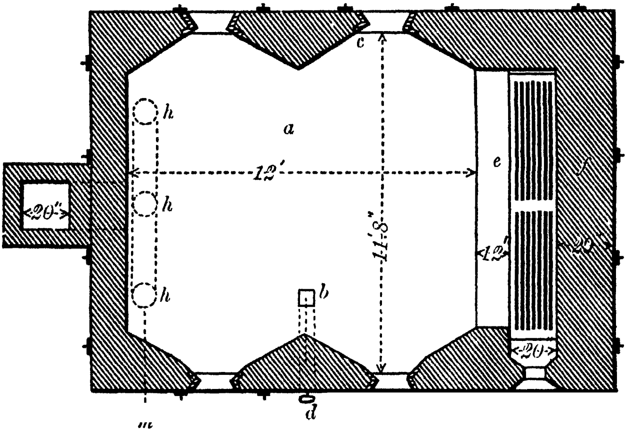

FIG. 80.—ROASTING FURNACE.

roasting furnace for one ton of sulphurets at a charge; *a* is the hearth bottom, about 12 ft. square. It is made of the hardest bricks, laid edgewise, close together, forming a stratum of 4 in. in thickness. There are four working doors, *c*, *c*, which enable the roaster to reach all points of the hearth conveniently with light rakes. In the middle of the length at the bottom, near to the doors toward the chlorination vats, is a square discharge-hole, *b*, which is kept shut by a slide, *d*, during the roasting.

Below the floor is an arch through the whole width of the furnace, through which the hole, *b*, passes. An iron car on rails receives the roasted ore when discharged, and wheels it to the cooling place.

The bridge, *e*, is from 10 to 12 in. wide, and from 8 to 10 in. high. It separates the hearth from the fire-place. If possible, it should be made of some refractory material, like fire-bricks.

The outside wall, *f*, is often made 12 in. thick, but it is always better to give sufficient substance to the wall on account of the heat, and at the same time to obtain a strong support for the roof; the thickness ought not therefore to be less than 24 in. Economy here would be misplaced.

The roof is generally 20 in. above the bottom, as the greatest distance; if it is less, although the form of the furnace is improved, it is less likely to be durable, unless the work is perfect. The length of the bricks gives the thickness of the arch, that is 8 in. This arch, if all the bricks are placed perpendicularly, will stand a great deal longer than one of 12 in. thickness, when the bricks are laid with the long sides alternately horizontal and perpendicular.

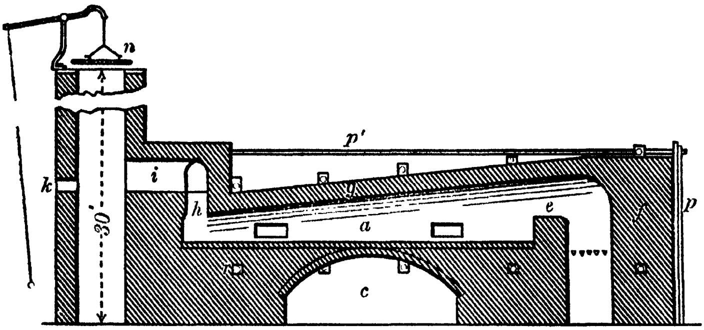

FIG. 81.—SECTION OF SINGLE FURNACE.

There are three circular openings, *h h*, in the roof, each 10 in. in diameter, communicating with the chimney by the flue, *i*, opposite which is a small door, *k*, for the purpose of cleaning the flue, *i*, from time to time. For the same purpose an opening must be prepared in front. The best way to regulate the draft is by means of a cover, *n*, on the top of the chimney; but when there are two or more furnaces to one chimney, a damper in the flue, *i*, will answer the purpose.

IRONS.—
Scale ⅜ inch
= 1 foot.

The sulphurets are charged through the roof by means of a hopper. It is of importance to secure the furnace against expansion by the use of grappling irons, of which there are eighteen.

The iron rods crosswise are from $\frac{6}{8}$ to $\frac{8}{8}$ of an inch in diameter; the others, *p*, which are placed over the length of the furnace, are stronger—being 1 in. The grappling irons (represented in Fig. 82) are made of cast iron, and are about 4 ft. long. For the passage of the rods of the lower ends, square holes, *r* (Fig. 81), are provided through the masonry of the furnace. Another kind of grappling irons is shown in Fig. 83. They are of wrought iron, and must be at least 1 in. thick. They are cheaper than those of cast iron, but the latter are preferable.

Double Furnaces.—This kind is represented by Fig. 84. The heat which escapes from the single furnaces is conveyed through the flue, *e*, over the upper hearth, *b*, of the same size, having the working door on the opposite side. On this second hearth the ore loses a great deal of its sulphur, and is drawn through a discharge hole in the middle of the floor on to the lower hearth, where the roasting is finished. From *b* the heat must pass a third hearth, *a*, before it enters the chimney. It is a drying rather than a roasting place, as the heat is very moderate. Some furnaces of this kind have an auxiliary fireplace for the second hearth, which, for the roasting of auriferous pyrites, is entirely superfluous, but would assist in a chloridizing roasting of silver ores.

FIG. 83.
GRAPPLING
IRONS.—
Scale ⅜ inch
=1 foot.

In Freiberg, Saxony, there were three double furnaces for the roasting of lead ores for smelting purposes. These three furnaces have been combined into one, as represented by Fig. 85. The heat passes the lower three hearths, and then, ascending through the flue, the upper ones. From the flue, *a*, it is conducted over the drying hearth, *b*, built in the same way as *a* in Fig. 84; from *b* the ore is drawn on to the next lower hearth, *c*, through *f*, and is removed after two hours to *d*, where it is again exposed to the heat for two hours.

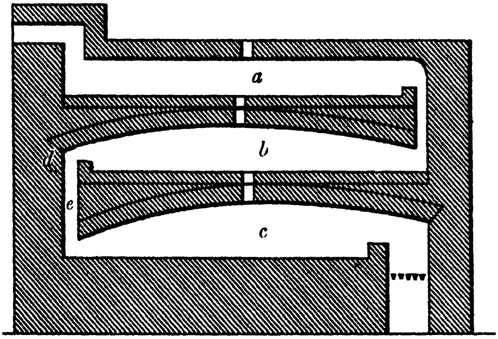

Fig. 84.—Double Furnace. Scale ⅛ inch = 1 foot.

Every two hours the ore advances a step till it arrives at *e*, where the strongest heat has now already a half-smelting effect upon it. Every two hours one charge of 800 lbs. of roasted ore is obtained from the hearth, *e*. The lower furnaces have two working doors each, the upper one three. There is not much stirring in this operation, and a kind of shovel is more in use than the hoe. The fuel used is coal.

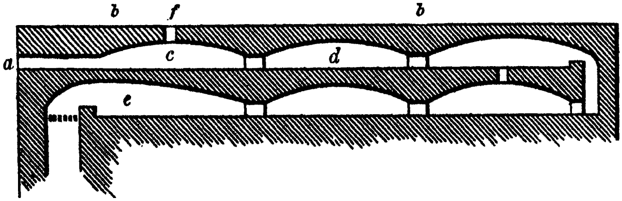

Fig. 85.—Double Furnace. Scale $\frac{1}{16}$ inch = 1 foot.

Pontgibaud Furnace.—A different construction is shown in Figs. 86 and 87. The first is the plan, the second a section, of a furnace built by Mr. Deetkin, in Grass Valley, for the sulphurets from the Eureka mine. The construction is like that of the roasting furnace for lead ores at Pontgibaud. The upper hearth is above the lower, and a continuation of it, interrupted by a step flue, *a*, of 7 ft. 10 in. in height—in the drawing, less. The upper hearth is 6 ft. wide by 39 ft. in length, furnished with working doors upon each side, twelve in number. The chimney, *b*, about 25 ft. high, is built after Deetkin's cheap plan, of 4 in. thickness, 28 in. each side in the clear.

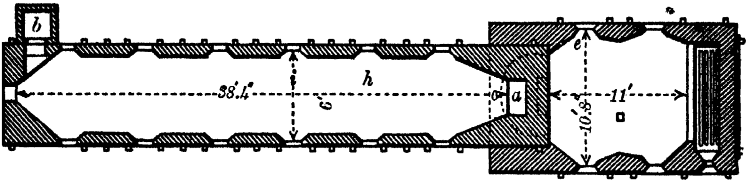

FIG. 86.—PLAN OF PONTGIBAUD FURNACE. Scale $\frac{1}{16}$ inch = 1 foot.

This kind of furnace requires more room than the double furnace (Fig. 84), but the work of stirring is less tiresome, as the roaster is not obliged to constantly step up and down.

FIG. 87.—SECTION OF PONTGIBAUD FURNACE. Scale $\frac{1}{16}$ inch = 1 foot.

Another advantage is the extent of the upper hearth, which receives 9 tons of ore without difficulty; whereas the charging of a three or four story high furnace is very troublesome unless favoured by a sloping locality. The rising flue, *a*, Fig. 87, is not necessary; on the contrary, it uselessly consumes a part of the heat. There is also no special advantage in making the second hearth narrower, for although the heat is more contracted, the length must increase for the same quantity of sulphurets.

A roasting furnace ought to consist of at least three hearths, constructed like those shown in Figs. 80, 81, but arranged similar to the lower tract of Fig. 87. By this means it is possible to draw every ten or twelve hours a ton of roasted sulphurets from the hearth near the fire-place. It takes about 24 hours to roast a ton of sulphurets. The only way of shortening the time is by the use of long furnaces, in which a great quantity of stuff is gradually prepared by being moved in successive portions towards the fire-place.

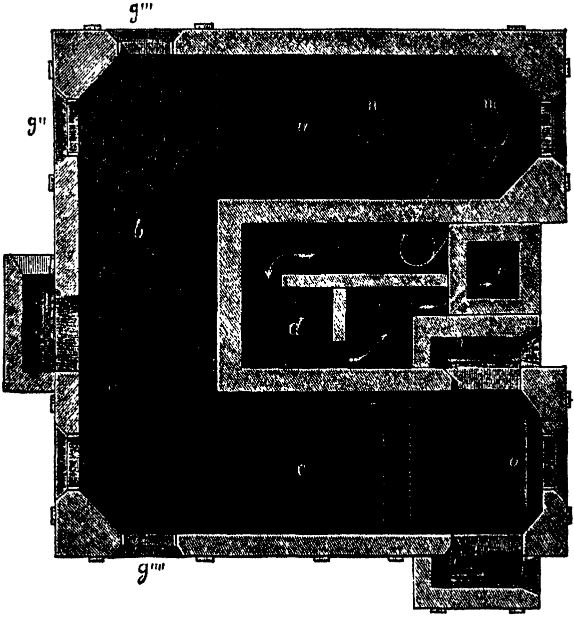

Fig. 88.—Küstel's Roasting Furnace.

Küstel's Roasting Furnace is shown in Fig. 88, which is on a scale of 8 ft. to 1 in. In this furnace there are two very important improvements. The first consists in breaking the straight line of the long furnace. The working doors, *g g*, are placed so that no lateral work is performed ; only drawing and pushing on an inclined hearth is required. The ore is introduced through the hopper, *n*, on the upper hearth, *a*, spread

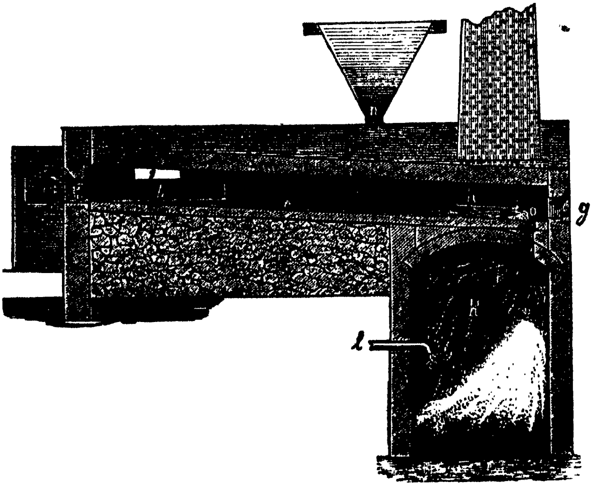

FIG. 89.—KÜSTEL'S ROASTING FURNACE.

equally, and after an hour's time drawn at *g''* and pushed from *g'* upon the second inclined hearth, *b*, and from this upon the third, *c*, in the same way. No stirring is required unless very difficult ore is under treatment. The moving of the ore from one end of the furnace to the other is generally sufficient. The necessary heat is kept up by two or three fire-places, *h h*. The gases pass through the flue, *m*, above the roof into the dust chamber, *d*, and escape through the chimney, *e*. The arrangement of having the working doors at the end of the long sides enables the roaster to do a great deal more work than if tired out by the old way of moving the ore toward the fire-bridge.

The other improvement is the chloridizing chamber, K, in Fig. 89. The purpose of this chamber is to shorten the time of roasting. It has been ascertained that the ore at rest in a red-hot condition continues to be chloridized when drawn out of the furnace. The ore falls through the opening, *o*, into the chamber, and remains there red-hot for two or four hours as may be required. Chlorine and volatile chloride metals, which are evolved, pass into the furnace and continue to chloridize the ore all along the furnace. In case there is no sulphur in the ore sulphurous acid gas can be introduced through the pipe, *l*, simply by burning sulphur. The sulphurous gas is transformed into sulphuric acid and liberates the chlorine from the salt.

The inventor claims that a furnace of this kind, by the aid of the chamber, K, can put through from 15 to 20 tons of ore in 24 hours, employing two shifts of three men each, consuming less than one-fifth of a cord of wood per ton of ore.

Bruckner's Cylinders.—Bruckner's revolving cylinders for roasting ores are used not only in the chloridizing roasting of silver ores, but also in the oxidizing roasting of iron pyrites which contain gold.

Fig. 90 is an elevation of the cylinder in perspective; Fig. 91 is a longitudinal, and Fig. 92 a transverse section. The exterior of the cylinder is a shell of boiler iron, 12 ft. long by 5 ft. 6 in. in diameter. The ends are partially closed with similar material, leaving in the centre a circular opening about 2 ft. in diameter, bounded by a flange projecting several inches. Upon one side is placed an opening closed by a hinged door. Upon the outside of the cylinder are bolted three bands, as shown in Fig. 90, in which the section of the first is square, and that of the third semicircular; the second or middle band is a strong spur gear. Passing through the cylinder are six pipes parallel to one another, in a plane at an angle of 15° to the axis of the cylinder; these pipes also lie in this plane at an angle of from 30° to 35° to the longitudinal axis of the plane, as shown in Fig. 91, where the internal arrangement of the cylinder is seen, a perforated diaphragm being formed through part of the cylinder by means of perforated plates placed between the pipes, the plates being held in place by longitudinal grooves upon these pipes.

The entire cylinder is lined with brick (common building

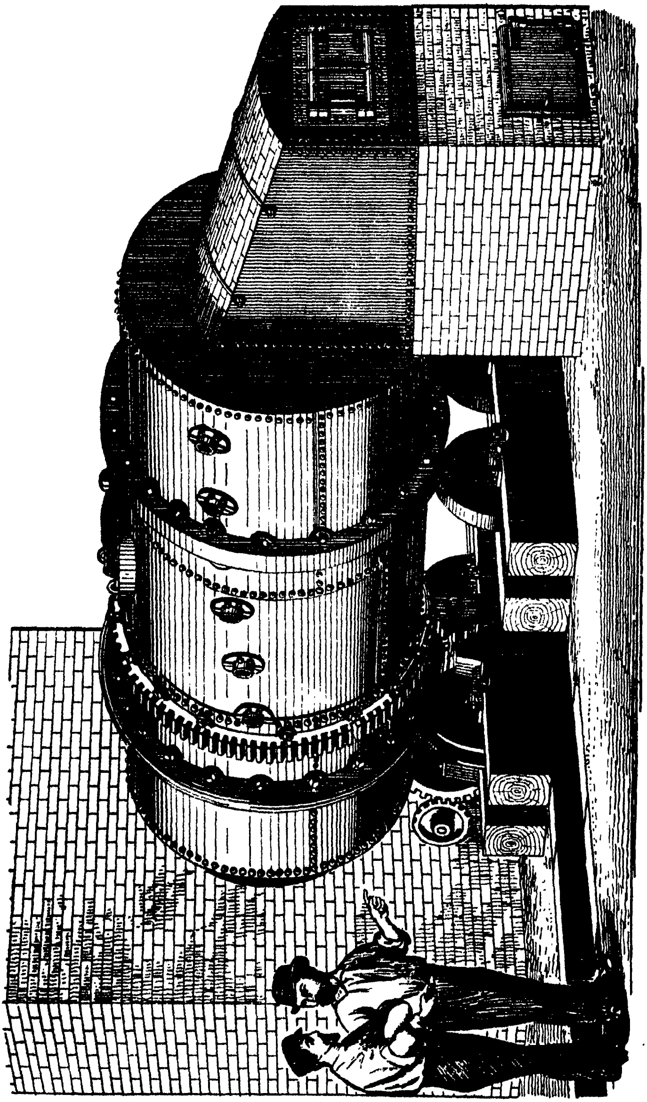

Fig. 90.—Bruckner's Cylinder.

bricks have been found to answer the purpose very well), the bricks being placed in the following manner:—The entire side of the cylinder is covered with one layer, laid flatwise, thus forming a lining about 2½ in. thick; there is an additional layer extending from each end of the cylinder about 15 in. to the centre of where the nearest pipe passes out; then additional concentric layers are added thereon, until the circle is contracted down to the size of the opening in the end, which is also lined, and each layer falls short of the preceding one by about 2 in., thus giving the end linings a conical form, the entire lining being laid in a mortar of one part fire-clay, two parts pulverised old fire-brick and water, all thoroughly mixed and beaten. The cylinder is supported upon four large friction rollers, two of which have a groove on their periphery, to loosely receive the semicircular band, and thus holding the cylinder longitudinally in place. The other two friction rollers are made without a groove, and bear upon the square band, thus accommodating themselves to the expansion and contraction of the cylinder, or any irregularities of form, all of which are seen in Fig. 90. Rotary motion is given to the cylinder by means of a pinion placed under the cylinder and gearing into the spur gear-band. Upon the other end of the pinion shaft are placed two bevel wheels, into which gear two match wheels, which latter are loose upon the driving shaft, standing at right angles to the pinion shaft. Either of those wheels can be attached to the driving shaft, thus communicating the speed of revolution of one or the other of the bevel gear as may be desired.

Inasmuch as by wear or settling the axis of the cylinder may possibly be thrown out of proper line, the following means of adjustment are provided, but not shown in any of the figures, viz:—Each journal box of the friction rollers is held in position by adjusting screws, by which it can be moved horizontally to or from the centre line of the machine, thus giving entire control of the lateral and perpendicular adjustment of the cylinder which they support.

The circular flange of one end of the cylinder loosely projects into a fire-box, best seen in section to the left of Fig. 91.

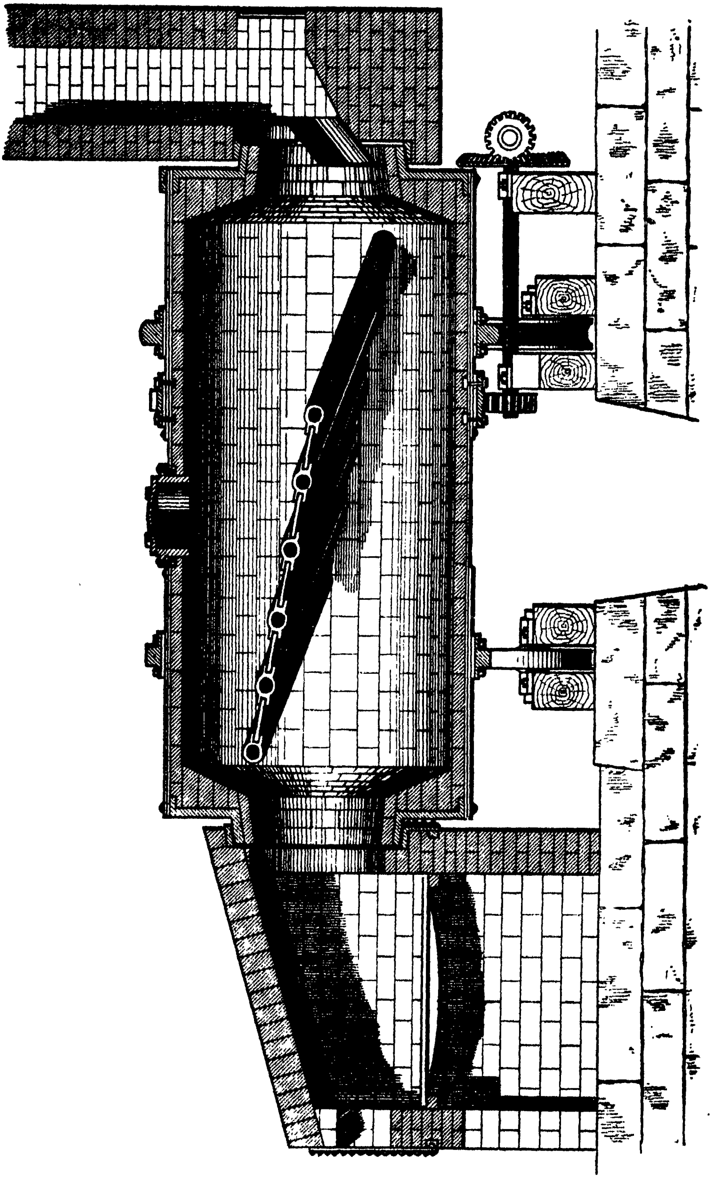

FIG. 91.—BRUCKNER'S CYLINDER. Longitudinal Section.

The other end projects into an opening communicating with dust chambers and a chimney. There is placed in the bottom of the flue a shoe projecting into the cylinder, which catches such dust as may fall back, and returns it into the cylinder in lieu of allowing it to escape through the crevice between the cylinder flange and opening into the flue. A door is placed in the flue opposite the opening, through which the interior of the cylinder and its contents can be readily examined at any time.

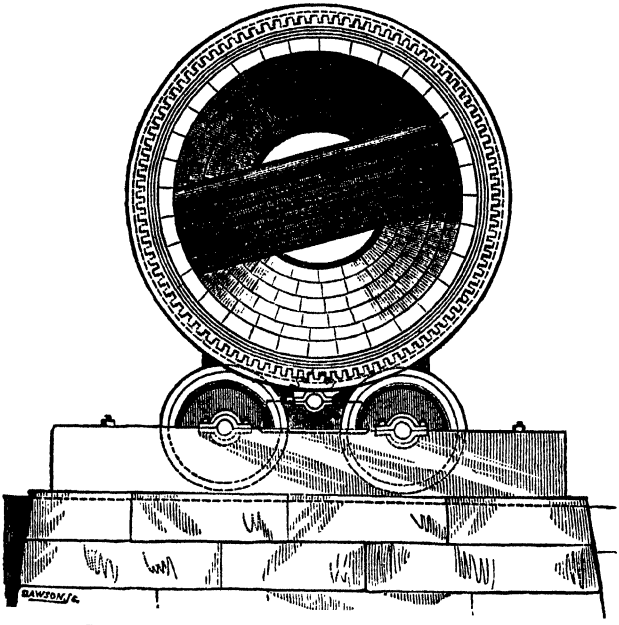

FIG. 92.—BRUCKNER'S CYLINDER. Transversal Section.

Bruckner's cylinder has been found to give excellent results in roasting the compound auriferous pyritic ores to be treated by the Plattner process, in which case a small quantity of charcoal is subsequently introduced to the charge, so as to facilitate the decomposition of the resultant sulphate of copper.

The Operation of Roasting.—Roasting the sulphurets or arseniurets is very simple, the principal aim being a perfect dead roasting—that is, the expulsion of all sulphur. After the furnace has been heated for some hours, the sulphurets are introduced through a hopper above the roof into the furnace, and spread over the hearth. A furnace such as shown in Fig. 80 takes one ton at a charge. One man is sufficient to attend the furnace. The fire is kept moderate, as the burning of the sulphur creates so much heat that nearly half of the sulphur contained is expelled thereby. On exposing a new surface of the mass (which in a short time becomes dark red hot) to the air, the burning sulphur having a bluish light can be seen distinctly. The hoe is principally used for stirring. It must be as light as possible, and is represented by Fig. 93. The plate, *a*, is made of boiler iron, 10 in. by 4½ and ⅛ of an in. thick, and riveted to the round iron rod, ⅝ of an in. in diameter, and 7 to 8 ft. long. One or two hoes are required on each side of the furnace.

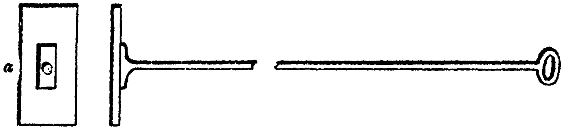

FIG. 93.—HOE.

The stirring is performed at intervals of about fifteen minutes. Whatever may be said about the uselessness of continual stirring, the fact must not be overlooked that sulphurets exposed to the air are sooner deprived of their sulphur than those which are nearest the hearth and excluded from undecomposed air. If this were not the case even a partial stirring would be useless. For that reason the intervals must be only of such length as is necessary for the relief of the roaster.

Proper attention should be paid to the regulation of the draught, to keep the quantity of air which enters the furnace in proportion to the heat; and while the sulphur is burning it is better to leave some of the working doors open, so as to supply plenty of air to the burning sulphur, but taking care that the heat is not lowered too much.

There is no advantage in having a very high heat at first, and it is better on single-hearth furnaces to roast at the beginning with a low heat and finish with a high heat; but a proper regulation must be maintained so as to avoid the balling of the ore, or melting into lumps.

The ore should be changed from the front part of the furnace to the back, and *vice versâ*, as there is necessarily more heat near the fire bridge, and this precaution will give the whole ore mass a chance to be exposed to the higher temperature of the furnace.

In proportion as the oxidation of the sulphur and iron approaches completion the temperature decreases, and it is necessary to use more fuel, so as to keep the mass at a good red heat. It takes from twenty to twenty-four hours before the roasting of one charge may be considered finished. If in throwing up the sulphurets in the furnace by means of the hoe or shovel many brilliant sparks should appear, this would indicate that the roasting has been carelessly performed, and it must be continued until this appearance ceases.

In a double furnace, Fig. 87, the heat in the lower hearth is always kept bright. One ton is roasted below, and about nine tons are spread on the upper long hearth. There are two roasters busy at the same time, one with the finishing, the other with the preparatory roasting, but they assist each other if needed. When one charge is drawn into the iron car below the hearth, the other load is shortly beforehand pushed down from the upper hearth, *c*, through the flue, *a*, and is now ready at *d* to be drawn into the furnace by means of hoes through the doors, *e e*. There is another door, *g*, behind the flue hearth for the same purpose. This charge is exposed to the preparatory roasting for about twenty-four hours on the upper hearth, and consists of a small part of undecomposed sulphurets and of oxides and sulphates. At a lively heat and with active stirring at intervals, all the base metals ought,

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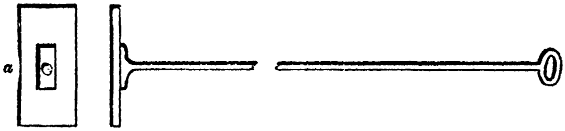

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In proportion as the oxidation of the sulphur and iron approaches completion the temperature decreases, and it is necessary to use more fuel, so as to keep the mass at a good red heat. It takes from twenty to twenty-four hours before the roasting of one charge may be considered finished. If in throwing up the sulphurets in the furnace by means of the hoe or shovel many brilliant sparks should appear, this would indicate that the roasting has been carelessly performed, and it must be continued until this appearance ceases.

In a double furnace, Fig. 87, the heat in the lower hearth is always kept bright. One ton is roasted below, and about nine tons are spread on the upper long hearth. There are two roasters busy at the same time, one with the finishing, the other with the preparatory roasting, but they assist each other if needed. When one charge is drawn into the iron car below the hearth, the other load is shortly beforehand pushed down from the upper hearth, *c*, through the flue, *a*, and is now ready at *d* to be drawn into the furnace by means of hoes through the doors, *e e*. There is another door, *g*, behind the flue hearth for the same purpose. This charge is exposed to the preparatory roasting for about twenty-four hours on the upper hearth, and consists of a small part of undecomposed sulphurets and of oxides and sulphates. At a lively heat and with active stirring at intervals, all the base metals ought, 4 to 5 in. in thickness. This filter remains always in the vat; the shovelling out of the residue, therefore, must be done carefully on approaching the filter bottom. There are two holes communicating with the space below the false bottom. One is for the reception of the lead pipe, *d*, by which the chlorine is introduced; the other is provided with a leaden cock, *e*, for the discharge of the lixivium. This discharge side of the vat stands ½ in. lower.

The wooden vat would absorb a great deal of the gold in solution, if the inside were not coated with some material which prevents the soaking in of the fluid. Mr. Deetken, who first introduced this process in California, uses one part pitch melted with one part of tar. This is a cheap and suitable mixture, and is applied hot by means of a brush. It is a matter of course, also, that the boards of the false bottom be coated carefully on all sides, as well as the whole of the inside.

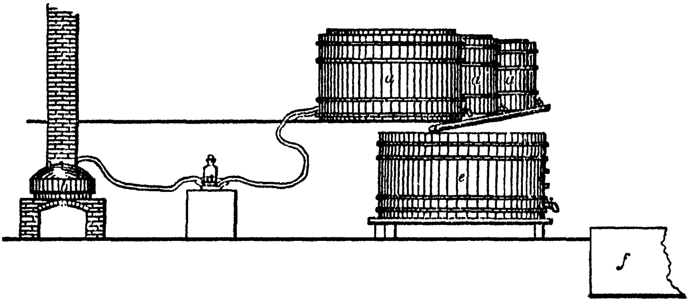


FIG. 95.—CHLORINATION APPARATUS.

There are also vats or tubs 5 ft. in diameter and 3 ft. high, holding 2 tons of sulphurets. Three or more of them are arranged in one row, as shown in Fig. 75, *a a*. They are conveniently managed, and are to be preferred where small charges of custom ore are to be treated. The usual charge of a vat like that shown in Fig. 94 is 6 tons of roasted sulphurets.

Chlorination vats with a greater diameter are preferable to higher tubs of the same capacity, for the reason that a low column of sulphurets assumes a less dense condition; and also because of the greater cubic contents of the free space above the sulphurets, which is filled with chlorine, so that an accidentally greater consumption of the gas can be replaced.

The cover, *g*, Fig. 94, must fit as well as possible in the step of the vat side, but not too tight; the planks, however, have to be fitted together tightly with tongue and groove. For the purpose of lifting, there are generally three or four chains fastened to the cover. Ropes are of no use, as they are destroyed in a short time by the gases. After the ore is sifted into the vat, and spread evenly, it is ready for chlorination.

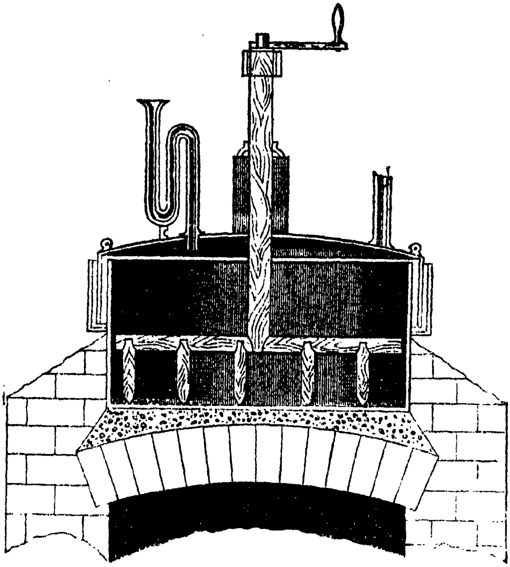
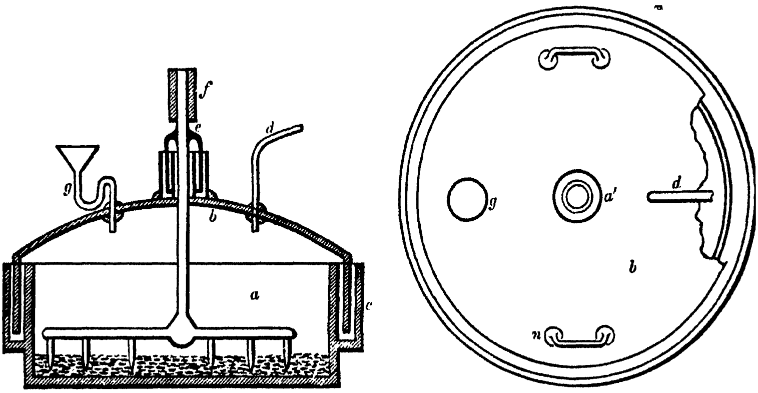


FIG. 96.—CHLORINE GENERATOR SET IN FURNACE.

Treatment of the Ore with Chlorine Gas.—The chlorine is produced in a leaden vessel, as shown in Figs. 96 and 97. The first (Fig. 97, A) is a vertical cross section, Fig. 97, B, a top view with the cover on. The circular tub, *a*, has an outer ring, *c*, 6 in. deep, for the reception of the ring-shaped side of the cover, *b*. A similar small ring, *a'*, is on the top of the cover, which receives the collar, *e*, fastened to the leaden stirrer, *f*. There is also a short leaden pipe, *g*, bent in the shape of the letter S, through which the sulphuric acid is introduced, the outer end forming a funnel for this purpose. Another lead pipe, *d*, conveys the chlorine to the vat. The cover is taken off, and for a charge of three tons of roasted sulphurets the ollowing materials are introduced: 30 lbs. of pulverised peroxide of manganese; 30 to 40 lbs. of common salt, according to quality; 75 lbs. of sulphuric acid, of 66 degrees; and 45 lbs. of water.



A. B.
Fig. 97.—Chlorine Generator. Scale ⅜ inch = 1 foot.

The water, salt, and manganese are introduced first, and the generator covered. The bottom is made of 16-lb. and the sides and cover of 8-lb. sheet lead. The two rings, *c* and ', are filled with water, and thereby the contents of the nerator shut up air tight, with the exception of the two lead es, *g* and *d*, of the cover. The gas generator stands over a ll furnace, as represented in Fig. 96. The sulphuric acid is introduced through the pipe *g*, Fig. 97, but not all at once. ; bottles are generally sufficient to create so much heat the development of the gas takes place in sufficient quantity. No fire is made as yet under the generator. The chlorine is not conveyed directly to the chlorination vats, but through a purifying apparatus, as represented in Fig. 98.

An ordinary wash basin, or some similar vessel, *b*, receives two ¾-in. lead pipes One of them, *a*, conveys the chlorine from the generator, and is bent a little upward. The other is bent in the same way, but stands higher. Both ends are covered with a bottle, the bottom of which is cut off. There is sufficient clean water in the dish to stand one half or three-quarters of an inch above the mouth of the pipe, *a*, so that all the gas which enters the space in the bottom is forced through the water, which takes up the muriatic acid. The chlorine then passes through the pipe, *a*″, which is as long as may be required by the distance of the vat, enters the space below the false bottom, and gradually permeates the ore. The water through which the gas passes absorbs, if cold, about two and a half volumes of the chlorine, and is then saturated, but is still good for the purpose of taking up muriatic acid. The warmer the water is the less chlorine is absorbed. It is therefore wrong to introduce a continual stream of cold water into the wash basin, *a*, as is sometimes done, as a good deal of the chlorine is thereby lost. The water in the basin may be renewed once or twice during the operation with warm water.

This apparatus is not only for the absorption of muriatic acid —since, if a portion of the acid should happen to enter the vat, and, forming sulphuretted hydrogen, precipitate metallic gold, this would be converted again into a chloride in presence of abundant chlorine—but the apparatus is an indispensable indicator of the progress in the gas generator. The bottle, *b*, Fig. 98, must show a greenish gas, and the bubbling from the pipe, *a*, must be very lively. If this should not be the case another bottle of sulphuric acid must be introduced, and the addition continued as often as the development of the gas becomes weaker. After the last bottle has been used up a moderate fire must be made below the gas generator. The arch, as shown in Fig. 76, is very flat, and only two inches thick in the middle. Care must be taken to have a half-inch layer of sand over the arch. An open crack would cause the melting of the bottom of the lead vessel. It is also necessary to turn the stirrer, *f*, now and then, carefully, to prevent the caking of the ingredients.

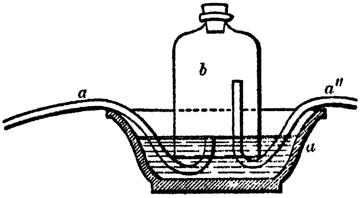


FIG. 98.—WASH BOTTLE.

The vat, after the ore has been sifted in, as before described, is left uncovered. It takes from three to six hours top of the charge. The progress can be easily watched by taking samples from underneath the surface. When the smell of the chlorine odour is perceived within a few inches of the surface of the charge, the cover is laid over the vat, and the edge, *i*, Fig. 94, all around the cover, luted with wheat flour dough. If there be any cracks in the cover they must be carefully pasted so that no chlorine can escape anywhere. The only opening not shut is a hole one inch in diameter, *k*; but as soon as the gas commences to escape, that is plugged up and secured with dough.

The chlorine is now permitted to operate on the gold for twelve or eighteen hours. If the vat holds five or six tons, the gas generator can be put in operation before the vat is entirely filled with ore, because the ascension of the chlorine is also slow. The pipe, *d*, is removed and the hole plugged up.

All the apparatus should be examined at intervals to see that there is no loss of gas, or at least whenever such loss may be perceived by the odour. For this purpose ammonia serves. A glass rod dipped into it and carried close around the place where the loss is indicated by the smell, will immediately give off white fumes when in contact with the chlorine, and show the place where it escapes. To prevent the drying of the dough around the cover it may be covered with strips of wet cloth.

For the production of chlorine the following formula is also used: 1 part manganese, 2 parts muriatic acid, 1 part sulphuric acid diluted with 1 part water.

Lixiviation.—After twelve, or if the sulphurets contain coarser gold, after fifteen to eighteen hours, the cover is taken off and water introduced. If it should happen in taking off the cover that no gas is found over the ore it will be advisable to shut the vat and to immediately impregnate the ore again with chlorine, as in nine cases out of ten the extraction of gold will fall short. This, however, with proper management, does not often occur.

Before taking off the cover from the vat, the plug can be removed from one of the vent-holes, and by holding a glass rod dipped in ammonia before the vent-hole, the evolution of fumes will indicate the presence of chlorine above the ore.

The water should flow in quickly, and in such a way as not to strike on one point, and thus produce a deep hole in the mass. The cock, *e*, is shut, and the water-flow continues until the surface of the charge is covered and no air bubbles appear. The water is then stopped, and the cock, *e*, opened. A small stream of water running into the vat must replace as much as flows through *e*, and so keep the surface of the ore always covered.

The respiration of the chlorine is injurious, and it is therefore advisable to avoid the inhaling of the gas as much as possible by leaving the room until the gas disappears. The best plan is an arrangement by which the gas can be carried out of the building; for instance, to put a movable wooden pipe 6 in. square about the proposed opening, *l*—or it may be more convenient to fix a lead pipe through the side of the vat, near the top, through which the gas will be forced by the entering water out of the building. In this case the water would have to be conveyed by india-rubber hose through the opening, *l*, of the same size as the hose. In order to prevent the stream of water making a hole into the ore, a perforated wooden distributer fixed to the cover, as shown in Fig. 94, would answer the purpose.

It is advisable to collect as much gold as possible in a small volume of water, and the best results are obtained by letting a shower of water percolate the ore. This, in travelling slowly through the ore mass, takes up nearly all the gold which permits of an easy precipitation. The gradual displacement of the surplus chlorine can be effected through a discharge pipe fixed in the top side of the vat and passing into another leaching vat, and this will allow of the operation being carried out while the cover is on the vat.

A trough below the cock, *e*, receives the solution and conveys it into the precipitating tub, *e*, Fig. 95. The trough must be lined with sheet-lead, avoiding sharp corners, or at least it must be well coated with tar and pitch in default of sheet lead. Great care must be taken to prevent the waste of the solution. Not a drop of it should be seen outside the trough.

The Precipitating Vat (*e*, Fig. 95) is a wooden tub like the chlorination vat (Fig. 94), but without a false bottom. The staves must fit together perfectly. A vat 4 ft. in diameter and 3 ft. high is sufficient to receive the solution of three tons of ore. Deetken's ten-foot vat, containing six tons of sulphurets, requires two precipitating vats, the one 5 ft. and the other 6 ft. in diameter, and each 2 ft. high. The vats ought to be lined with sheet lead. A better shape would be a rectangular box, having a half-round somewhat inclined sheet-lead bottom, as this would permit easier and better cleaning. In default of a leaden lining, the vats must be coated with a mixture of pitch and tar, otherwise the wood absorbs some of the gold solution. A better and smoother coating is obtained by the use of the so-called "asphaltum cement," which should be applied twice before it is ready for use, as the cement is too liquid for a single coating. A very smooth surface of the vat is important, else it is difficult to gather all the finely precipitated gold.

From time to time samples are taken in a clean white glass tumbler from the solution at the end of the trough, when it is observed whether an addition of a clear solution of sulphate of iron (green vitriol) causes a dark precipitate. If the solution after the addition of the precipitant should remain perfectly clear, the water supply in the chlorination vat must be stopped, and all the liquid contents of the vat permitted to flow into the precipitating vat.

Precipitation.—The precipitant for the gold is a solution of sulphate of iron. It is known also under the name of "copperas," or green vitriol, and is in light green crystals. Dissolved in water it generally makes a muddy solution, and deposits light sediment, which must not be disturbed in drawing off the clear solution from above it. For this purpose a leaden syphon will answer, but it is, however, better economy to prepare the precipitant fresh in the chlorination works. In a barrel or tub of about ten cubic feet contents are put 50 to 60 lbs. of pieces of old wrought iron; then five or six buckets of water are added, with 20 to 30 lbs. of sulphuric acid. This is prepared two or three days before the solution is required for use. One or two buckets of this solution must be poured into the precipitating vats before the gold solution is allowed to flow in, so that the precipitation may begin immediately. After this is added as much of the precipitant as is required, which can be ascertained by taking a sample out of the precipitating vat, passing it through filtering paper, and mixing it with the precipitant. If after a time the mixture should darken, a little more of the iron solution must be introduced into the precipitating vat. The precipitating gold requires some time before it is all deposited on the bottom. The fluid must appear perfectly clear before the water can be drawn off. Generally the mixture stands undisturbed all night. The upper plug of *e*, Fig. 95, is removed, and the clear liquid conveyed into another vessel, *f*, of sufficient capacity, till it is nearly all run out of the precipitating tub through all the plug holes. The discharge must be performed carefully, so that the flow shall be always clear.

It is better if several chlorinations have been performed before the gold is taken out, as with a large quantity of gold there is a less percentage of loss by wastage. The gold is taken out carefully by means of a dipper or scoop and put into a clean porcelain dish or enamelled vessel, and the rest is washed out through the lowest cock. It is well to run a jet of water over the sides and bottom, to wash off all the precipitated gold.

The gold obtained is then introduced into a paper filter, and subsequently dried in an iron porcelain vessel in a warm place or over fire. For melting purposes black-lead crucibles are less suitable than "Hessian" or clay crucibles; from the latter a purer gold is obtained. A little salt, some borax, and saltpetre are added as fluxes.

Zinc, antimony, and lead are not obstructive to the extraction of gold by chlorination, but it has not yet been determined by experience whether or not a great amount of galena would to any extent prevent chlorination. In all cases the presence of galena necessitates a good roasting and a strong finishing heat, in order to decompose the sulphates as much as possible. If the roasting is not properly finished, the undecomposed sulphurets and sulphates will absorb a great deal of chlorine, and the chloride of lead and antimony formed will be carried into the precipitating vat. Both are precipitated as a white powder (as chlorides) by dilution with the leaching water. It often occurs that such ore is accompanied by lime or calcareous spar; in this case the lixivium will contain chloride of lime, which is precipitated with the gold as gypsum, unless precipitated beforehand and separated by the addition of sulphuric acid. Heavy spar will probably behave like calcareous spar, and be precipitated by sulphuric acid.

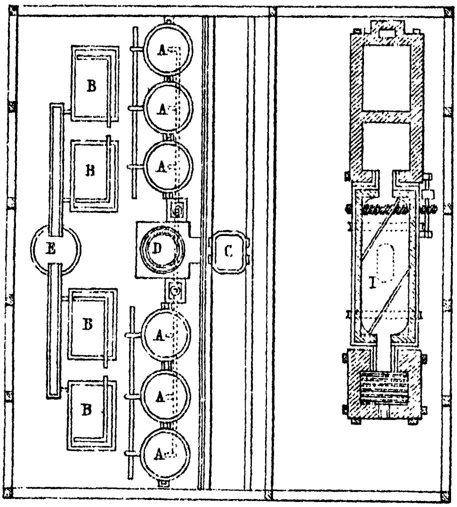


FIG. 99.—PLAN OF CHLORINATION WORKS.

The question in regard to the utilisation of the chlorine which remains in the vat after the chlorination of the gold has been finished must be decided by a practical trial. A vat of 7 ft. diameter and 2 ft. high, when filled with roasted ore to within 6 in. of the top, leaves about 47 cubic feet of space for the chlorine, as the moistened stuff contains more than 50 per cent. of interstitial space. Provided the roasting be well performed in auriferous iron pyrites, the whole free space in the vat will be filled with chlorine after the process of chlorination is finished. This ought to be the case under all circumstances; and if the roasted stuff consumes more gas than usual, it must be replaced by an additional quantity. Having then two chlorination vats, the communication between them for the purpose of conveying the chlorine from one to the other is easily obtained by the lead pipes, each leading from the upper part of one to the empty space (*c*, Fig. 95) of the other, the pipes being provided with cocks. As soon as the water is admitted through the hose into the vat, the chlorine will escape through the lead pipe into the other already prepared vat. It will be necessary then to fix a glass tube, bent in the shape of a horse-shoe, into the upper part of the vat, so that the height of the water above the ore can be seen and regulated accordingly. The chlorine transferred into the other vat will require a certain additional quantity from the generator.

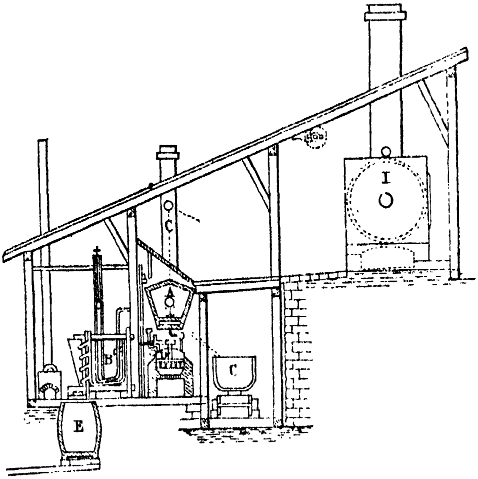


FIG. 100.—SECTION OF CHLORINATION WORKS.

Chlorination Works.—Their general arrangement is shown in plan and elevation in Figs. 99 and 100. 1 is a

Brückner cylinder, in which the ore undergoes a preliminary roasting to remove such precipitating agents as sulphur, antimony, and arsenic. The leaching vats are at A, the precipitating vats at B, trucks to remove spent ore at C, the gas generator at D, and the waste tub, where the water runs through sawdust, at E.

Cost of Treatment by Plattner's Method.—The expenses of working by this method are rather high, especially where the ordinary reverberating furnaces are still in use.

At a small 8-stamp mill in Amadar county, California, the superintendent informed me that the ore which is run from the battery over copper plates is concentrated, the sulphurets having an average value of £22 per ton. Their treatment cost £4 per ton, leaving £18 profit. The average proportion of sulphurets per ton of ore was 5 per cent., and the yield of gold obtained from the sulphurets about £1 per ton of ore.

Loss of Gold in Chloridizing Roasting by Volatilization.—Mr. C. A. Stetefeldt, an eminent American engineer, has investigated the subject of the loss by volatilization of gold during chloridizing roasting, a matter of great importance when treating auriferous silver ores. He found that the loss in gold in some Mexican ores, when roasted in a reverberatory furnace, amounted from 53 to 88 per cent.

He says that there is no doubt that the volatilization of the gold takes place with that of the copper chlorides, in case of coppery ores. The loss increased with the quantity of these chlorides formed and then volatilized. It is therefore essential during roasting to avoid the formation of a large quantity of copper chlorides. Temperature and time of roasting must also influence the result. Against this, however, he quotes an instance where 80 per cent. of the silver and from 68 to 85 per cent. of the gold were lost in chloridizing roasting with ores which did not contain copper.

Plattner speaks of the loss of gold in oxidizing roasting, but entirely neglects chloridizing roasting. G. Küstel records a loss of 8 per cent. of gold in roasting telluride ores with salt, and states that by increasing the temperature and time of roasting the loss in gold may be 20 per cent. and more. Mr. Aaron suffered heavy losses in the chloridizing roasting of gold-bearing sulphurets. In Australia, the chloridizing roasting of gold ores was abandoned on account of heavy losses.

A very large portion of the volatilized metal should be regained with proper condensation, and considerable attention is being paid now to the question of condensing fumes from roasting and smelting furnaces. Experiments on the electrical collection of metallic fumes and dust, which have been made by Mr. A. O. Walker in England, point to a successful solution of the problem.

Proposed Condensation of Fumes by Electricity.—Mr. Walker proposes to lead the fumes through dust chambers or flues, as is usual with smelting or roasting furnaces. He collects the fine particles of metals suspended in the fumes by means of a discharge of high potential electricity from metal points or edges, or other projections, situate in the flue passage chamber or other receptacle, and so placed that the current of air or gas containing the finely divided material or metallic vapour is carried or passes in close proximity to the discharging points. The discharge from the points electrifies the air or gas, and the charged air or gas then acts on the finely divided matter in it, causing it to cohere, condense, and deposit.

The separation and collection of metalliferous particles from the fume from lead-smelting furnaces is effected as follows. The flues from such furnaces are at present frequently conducted for long distances in a horizontal or inclined direction, in order that the fume may deposit a portion of the metalliferous matter which it contains. The flues may be much shorter than at present, and nevertheless a better deposit will be obtained, and the process may be applied in other receptacles or reservoirs.

At each point where the treatment is to be applied, within the flue or chamber a metal conductor is provided, which may be a rod, or combination of rods, spheres, plates, or any other convenient form of such dimensions as will be most suitable to the size of the passage or receptacle. In preference, two rods in the form of a cross—each arm of which occupies respectively about two-thirds of the height and breadth of the passage—can be used.

The surface of this conductor is studded all over with metal points or other projections—say at distances of two or three inches apart, and projecting (it may be) two or three inches from the surface of the metal conductor. This conductor is very carefully insulated in some suitable manner. It may, for example, be done by supporting it at the end of a horizontal metallic rod passing through the wall of the flue. An earthenware tube may be inserted into the wall of the flue, and the supporting rod passed through the tube, but without being in contact with it. Outside the flue the rod may be carried on glass legs, entering without contact at the neck into glass vessels, in which a small quantity of strong sulphuric acid is contained. This rod is connected with one terminal of any kind of electrical machine capable of developing high tension electricity by induction or by friction, the other terminal being connected to the outside of the flue or chamber or to earth. Or the electricity may be supplied at high potential by any other known means. The whole is so arranged that a continuous electric discharge is maintained from the points which stud the surface of the conductor within the flue or chamber into the air. Other means may be devised to insulate in like manner the sphere or conductor. Several of these arrangements may be applied along the flue at distances of a few yards.

The metalliferous particles will by this treatment be caused to agglomerate, and will be deposited in the portion of the flue beyond and near the place or places where they are submitted to the electric discharge. The deposit may be removed mechanically from the flue from time to time, or it may be removed in part by a stream of water flowing along the lower part of the flue, as is the practice at present in flues for condensing lead fume, or in any other convenient way.

Mills and Chlorination Works at Plymouth Mine, California.—In an interesting account of the operations at these works, which was given by Mr. G. W. Small in a paper read before the American Institute of Mining Engineers, it was shown that if a mine is located near to centres of supplies, where fuel, labour, and chemicals can be cheaply obtained, the Plattner process can be carried out at a cost of about £2 per ton after the concentrates are delivered at the works. Few mines, however, are so centrally located as those he described, and the cost stated above—£4 per ton—should be taken as a *low average.* Mr. Small's report was as follows:—

"The ore, as it is raised from the mine, has an average assay-value of $11 per ton, chiefly in the form of free gold. All the ore goes directly to the stamp-mills, of which there are two. The older and larger mill contains sixteen batteries of five stamps each, with one Frue vanner to each battery. The new mill has eight batteries of five stamps and two Frues to each battery. The large mill is driven by Leffel turbine wheels, with a pressure of 80 feet, and a consumption of 600 miners' inches of water. The smaller mill is driven by "hurdy-gurdy" wheels, with a pressure of about 550 feet and a consumption of 150 inches of water.

"At both mills the tailings from the stamps pass over about 20 feet of plates on their way to the Frues. In each set of plates the first or upper one is copper, the rest are so-called silver plates. The bullion from the stamps is about 800 fine in gold and 200 in silver.

"The concentrates from the Frues average from 1¼ to 1½ per cent. of the ore stamped. They very rarely exceed 2 per cent. I was unable to get the exact assay-value of the concentrates, but it is said to vary between $100 and $200 per ton. The concentrates are treated at the chlorination works at the rate of one hundred tons per month. The capacity of the works is somewhat greater than this, but as the supply of concentrates is limited it is not deemed advisable to work them up any faster Care is taken to keep the concentrates always damp until they are put into the roasting furnace. If this is not done, a decomposition of the pyrites begins, forming lumps which do not roast, and which consequently cause a loss of gold in the residues from leaching.

"A *Fortschaufelungsofen** is used for roasting. Its dimensions, including fire-box, are 12′ × 80′. The hearth is one continuous plane, but the charges, of which there are three in the furnace at one time, are kept entirely separate. The furnace-men called the three compartments, the "drying," the "burning," and the "cooking" compartments. In the middle, or "burning" compartment, the ore is spread out very thin, and occupies about double the space of either of the other compartments.

"The furnace is worked by eight-hour shifts, one man on each shift, and one charge is drawn and a new one added in each shift. The charges weigh 2,400 pounds, and carry about 10 per cent. of moisture. The ore averages about 20 per cent. in sulphur, and just before the sulphur ceases flaming (in the second division of the furnace) 18 pounds, or ¾ per cent. of salt is added to the charge.

"The roasted ore from each shift is kept by itself on the cooling-floor until a tankful (about 4 tons) has accumulated from a single man's shift; then that lot is worked by itself. This enables the person in charge the better to control the roasting; for if only one lot out of the three is bad, it is presumable that the fault lies with the workman; but if all three are bad, the probabilities are that there has been a material change in the character of the ore, and the roasting process must be altered accordingly.

"The vats for chloridizing the roasted ore are 9 feet in diameter by 3 feet in height, and are four in number. They are slightly inclined, so that they will drain completely. The bottom of each tank is occupied by a filter about 6 inches thick, composed as follows: Light strips of three-quarter inch wood are first laid in the bottom of the tank at intervals of about one foot. Across these strips are laid six-inch boards, leaving cracks of an inch or more between the boards. On top of this loose floor are placed coarse lumps of quartz, and on top of this again finer quartz material, until a total depth of about five or six inches is obtained. Finally, this "sand-filter" is covered by another loose floor, the boards lying crosswise to the loose floor beneath, and pretty close together. This upper floor is intended merely to furnish a shovelling surface, so as to permit the removal of the leached ore from the tanks without disturbing the filter.

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* *Anglicè*, a long continuous reverberatory furnace.

"The ore to be chloridized must be damp (about six per cent. moisture). The working test is, to take a handful of the ore and squeeze it, then open the hand, and if the lump immediately begins to crumble and fall apart (not run) the ore has the requisite amount of moisture. The damp ore is screened into the tanks, so that it will lie as loosely as possible and facilitate the penetration of the chlorine gas. A coarse screen of one-half inch mesh is used for this purpose. The tanks are only filled up to within about three inches of the top. This is to insure that the entire contents of the tank are covered by water in the subsequent leaching, otherwise there will be great difficulty in washing out all the soluble gold.

"As soon as the tanks are filled as stated, they are ready for the introduction of the chlorine gas. This is introduced into the bottom of the tank from two opposite sides, and is continued until ammonia held over the ore gives off dense fumes of ammonium chloride. This usually takes about four hours. When this point is reached, covers are placed on the tanks, and the cracks are luted with a mixture of leached ore, bran, and water. The gas-generators, of which there are two employed at one time in charging a tank, are allowed to work on until they are exhausted; then they are disconnected and the holes in the tank are plugged.

"The tank is usually charged with gas in the morning, and is left standing for two days. On the third day the ore is leached. The tank is first filled with water, and allowed to stand a few minutes so that the water may penetrate all the ore. If no more water is absorbed, the liquor is drawn off at the bottom, care being taken to keep the tank full of water during the entire operation, which takes from four to five hours. In charging the tank, a gunny-sack is laid on top of the ore, where the wash water is afterwards to be introduced, in order to better distribute the water in the tank and prevent its washing and packing the ore.

"The liquor from the leaching-vats is conducted to settling-or storage-tanks, and about 40 pounds of sulphuric acid (66° B.) is added. (Experience has shown this addition of acid to be advantageous in obtaining a clean product in the subsequent precipitation. The chemical reaction is, however, by no means clear.) It is usually allowed to stand for twenty-four hours, but two hours are quite sufficient. It is then run into precipitating-tanks, and the gold is precipitated by a solution of sulphate of iron. The iron solution is added until, after stirring, a further addition produces no purple colour. After the gold is precipitated it is allowed to stand two or, if convenient, three days to settle; then the supernatant liquor is drawn off with syphons into a second settling-tank, where any gold that may have been drawn of by the syphons has a second opportunity to settle. The liquor stands in this tank until it is necessary to run it off to make room for another charge. Very little gold is found in this tank, and it is therefore only cleaned out once during the year. In the meantime, fresh liquor has been run into the precipitating-tanks upon the gold already precipitated there. In this way the gold is allowed to accumulate until the semi-monthly clean-up. Except when it is necessary to have them open, the precipitating tanks are kept covered and locked.

"In making the clean-up, the supernatant liquor is syphoned off, the gold gathered up and placed in a filter of punched iron lined with a sheet of ordinary filter-paper, and washed with water until all the acid and iron salts are removed. It is then dried, melted in crucibles, and cast into bars.

"The works extract from 95 to 96 per cent. of the assay-value of the concentrated sulphides. Two men, on day-shift, attend to all the work of handling the ore after it is washed (the leaching, etc.). The head man receives $3, the other $2.50 per day. Owing to the limited amount of ore allotted to the works, only three tankfuls are leached every four days. The men, however, are employed steadily.

"The sulphate of iron is manufactured on the spot. For this purpose an ordinary wooden tank about 4 feet by 4½ feet, standing outside the building in the open air, is used. The tank is kept full of water and supplied with old scrap-iron *ad libitum*, and for each charge to be precipitated about 40 pounds of acid are added to the tank.

"The precipitating-tanks, which are of wood, are protected from the action of the acids by a coating of "paraffin paint."

"I append an itemized statement of the cost of handling the ore. The basis of figuring is 100 tons of ore per month of 30 days. Consumption of chemicals in the leaching department, 24 days in each month:

Roasting:		
Three men, at $2·50 per day, for 30 days	$225 00	
1¾ cords wood at $4·25 ,, ,,	223 13	
54 lbs. salt at ¾ cent ,, ,,	12 15	
		$460 28
Generator: The charge is manganese, 30 lbs., salt, 34 lbs., sulphuric acid, 60 lbs.; therefore, for two generators:		
Manganese, 60 lbs. per day, 24 days, at $47 per ton	$33 84	
Salt, 68 lbs. per day, 24 days, at $15 per ton	12 24	
Acid, 120 lbs. per day, 24 days, at $60 per ton	86 40	
		132 48
Acid for settling-tanks (40 lbs.), and for sulphate of iron manufacture (40 lbs.), 24 days		57 60
Wages of leachers, at $5·50, for 30 days		165 00
Salary of foreman		125 00
Total		$940 36

Or, per ton of concentrates, $9·40 3/10, or £2 sterling."

In the next chapter will be found an account of several improved processes for the treatment of auriferous ores by chlorination.

CHAPTER VII.

LATER PROCESSES OF CHLORINATION.

THE MEARS CHLORINATION PROCESS—Treatment of the Arsenical Ores of the Deloro Mine—The Adolph Thies Process—The Newbery-Vautin Process—The Swedish or Munktell Process—The Ottokar-Hoffman Process—The Ottokar-Hoffman Furnace—The Von Pateras and Roeszner Process—Kiss's Method: how Distinguished—Cassel's Process for Electro-Chlorination of Gold Ores—Hauch's Treatment of Telluride Gold Ores.

The Mears Chlorination Process.—B. Howell Mears, M.D., of Philadelphia, when experimenting a few years ago upon roasted pyrites, with a small apparatus in his laboratory at home, met with an accident through the clogging of one of the outlet pipes of the vessel in which he was exposing some ore to the action of chlorine. The stoppage caused a pressure of gas in the vessel, and an explosion followed by which the whole thing was shattered. As the exposure of pulp to gas had been only for a few minutes, he thought to try the thing over again, and for that purpose gathered up the sand, broken glass, and debris; but, as it was inconvenient from lack of apparatus to immediately expose the pulp again, he tested the pulp to ascertain if any solution had taken place, and to his astonishment he found that the action had been about as thorough and complete as he would have expected to find it after hours of simply passing the gas into and through the moistened pulp. This was a discovery, and on it is based the now perfect process called after the distinguished discoverer.

In the Mears process the roasted ore is moistened with a small quantity of water, then charged into an iron lead-lined barrel which revolves, and into which gas is forced.

By this means the ore is kept constantly stirred and tumbled about. Several advantages are gained by this mechanical attrition.

(1.) The ore is kept lively, and every particle of it exposed to the action of the chlorine.

(2.) If there be any large, coarse grains of gold present, instead of a coating of chloride of gold forming around them, and thereby preventing further action of the gas upon the gold grains—as is inevitably the case where the pulp lies still in the vat—the chloride of gold, as soon and as constantly as it is formed, is removed, thus keeping up a continuous action of the gas upon the gold, and consequently rendering the dissolving of a grain of gold as certain and, within reasonable limits, as speedy as if all the gold were of the finest powder. This is a very important consideration, for I am convinced that in many cases where sulphurets are treated by the Plattner process, and poor results or high tailing assays obtained, it is because the gold-particles are so coarse as not to be entirely dissolved during the time allowed for chlorination—in other words, that the solution has been hindered by the protecting coat of terchloride of gold enclosing the gold particles and stopping further solution.

(3.) In California a good deal of the gold in sulphurets, or rather many kinds of sulphurets carrying gold and silver, have the gold so highly alloyed with silver that it is not easily dissolved by chlorine. It is almost impossible to dissolve this alloy of gold and silver contained in some sulphurets, even in nitri-muriatic acid, unless the coatings formed on the gold particles be constantly removed by rubbing. Part of the silver becomes converted into a chloride, coats the particles of gold, and retards further action of the chlorine. But this only occurs in an alloy having a certain proportion of silver to gold. It is not always so, but it is common. Roasted sulphurets carrying this kind of gold alloy, when treated with chlorine, present no difficulties in a revolving barrel, because by the mechanical attrition the coating is rubbed off by the sharp grains of sand as fast as it is formed, in a manner similar to the terchloride of gold previously spoken of.

The above advantages are undoubtedly accomplished by attrition, and are easily and speedily effected in the revolving barrel.

The pressure of gas which is applied while the barrel is revolving is gained by pumping in the gas (by means of a peculiarly but simply-constructed pump) directly through one trunnion of the barrel. The effect of this pressure is also to hasten the action. When the charge has been revolved enough the barrel is filled with water, thus liquefying the pulp, and the contents of the barrel are discharged into a filtering vat, through which the solution runs into precipitating tubs, where the gold is thrown down in the same manner as in the Plattner process. Another plan is to run the solution through tanks packed with pulverised charcoal, which of itself precipitates and catches all the gold from the solution, and, after a time, the charcoal is burned and the gold collected. The precipitation by a freshly prepared solution of sulphate of iron is preferred.

Several plants embodying the Mears process are now in successful operation in California, and the process itself has led to further developments, as will appear in the following pages.

Treatment of Arsenical Ores at the Deloro Mine.—Mr. Rothwell gives an account of the treatment of these ores, in which the Mears process is adopted for chlorination. The ores are gold-bearing arsenical sulphurets of iron (mispickel), carrying the theoretical proportions of 42 per cent. of arsenic, 20 per cent. of sulphur, and the remainder iron. The gangue is quartz, calc spar, and some talcose, slaty rock, evidently resulting from the decomposition of the wall rock, which on each side of the veins is syenitic granite. The treatment of the ore consists of six processes.

Crushing.—The ore as it comes from the mines is dumped over grate bars at the top of the mill building; what is too coarse to go into the rock breaker is broken by hand, and all then goes through the No. 1 rock breaker, which breaks it to a maximum size of 1¾ inches, then over fixed grate bars, which take out all pieces less than half an inch in size. The coarse then goes through two small rock breakers which crush it to a maximum size of about three-quarters of an inch.

Concentrating.—The peculiarity of the Deloro is, that the gold is contained for the most part in the mispickel (which carries when closely concentrated nearly $100, or £20, per ton as an average), and this mispickel is much more friable than the associated quartz and calc spar, which contain but small quantities of gold. The consequence is that, when the rock as it comes from the mine is coarsely crushed, we find the fine is composed, for the most part, of mispickel, and the coarse is quartz with a little mispickel. Taking advantage of this peculiarity of the ore, the mill was constructed so that the ore from the second crushers and the fine from the first crushers would go into the No. 1 revolving screen, 20 ft. long by 5 ft. diameter, where it would be sized into *fine* (*i.e.* passing through a mesh of less than an eighth of an inch), and the remainder into two sizes, which would go to coarse jigs, which are intended to separate into a rich and a poor product—the rich after drying joining the fine from the No. 1 screen, and going to the rich rolls to be crushed to the roasting size, and the poor going to the poor rolls and jigs. The object sought to be attained by this arrangement was the partial concentration of the ore without the usual loss in concentration; but as the roasting and chlorinating capacity of the Deloro works is yet far inferior to the mill capacity, it has been deemed wiser for the present to crush in the poor rolls all the ore as it comes from the second rock breakers, and concentrate it in the jigs. The crushing is done in Cornish rolls 36 in. diameter, 15 in. face, steel shells on double cone centres.

The ore, after passing through the rolls, is elevated to the two No. 2 dry screens, which have a length of 8 ft. by 4 ft. diameter. In these the ore is sized. All passing through a 1⅓ millimeter, which comes out of the end of the No. 2

screens, drops back into the rolls and goes through again. The fine ore is sized in the usual manner, except that the screens are much larger than usual, for the limiting capacity of such works is always found in the screens. Each set of rolls has a capacity of fully five or six tons an hour in grinding hard quartz and mispickel down from say 1 inch to $\frac{1}{16}$th of an inch.

The crushing (and consequently the screening in the No. 1 and No. 2) is dry. The sized ore is concentrated in ordinary Hartz jigs discharging through the bottom, and the slimes in Spitzkasten and Rittinger's tables. Mr. Rothwell says that his own preference would be for other concentrators of greater capacity than these tables, although they are found to do very fair work.

Roasting.—The concentrates are taken from the jig room in a tram-car, which, going up an inclined plane, delivers them directly into a hopper over the drying furnace. This is an inclined revolving cylinder* 20 ft. long, 36 in. diameter at small end, and 48 in. at large; and it has a conical addition of 24 in. in length at the small end, making a total length of 22 ft. The fire passes through this cylinder, and the capacity has never been tested to anything like its limit; but no doubt it would dry two tons an hour, if required. It is very economical in fuel.

As the dry ore drops out of the drying furnace, it is continuously raised by an elevator into No. 1 roasting furnace. This is a revolving cylinder 30 ft. long, 60 in. diameter outside, lined with 4½ in. of fire-brick, and with eight shelves through nearly from end to end. These shelves are formed of key-bricks 9 in. long, so that they stand 4½ in. out from the lining. In this furnace the arsenic and the greater part of the sulphur are volatilized, and pass out through a long series of arsenic condensing chambers, and through a centrifugal (Guibal) fan, 8 ft. diameter, 3 in. face, used to make the draft to the stack. The ore runs from the first cylinder through a pipe directly into the second cylinder, 20 ft. long, by 48 in., with a 4½ in. lining and six shelves, where the roast is completed. The escaping gases pass to a stack, which also serves the drying furnace.

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* For a description of the revolving cylinder the reader is referred to my "Metallurgy of Silver."

The air which feeds the No. 1 furnace is preheated by the escaping gases of the second cylinder, by passing through an air space between the two arches which form the top of the second roaster dust chamber.

The two roasting cylinders are jacketed, first with an air space, and then with a covering of mineral wool, and paper over that. The whole arrangement of the roasting cylinders, their jacketing, and the plan of utilising the escaping gases to heat the feed air for the first cylinder, are believed to be new, and are found very economical and efficient in practice.

The ability to make a sweet roast (such as is required for chlorinating) in a single operation, in continuous revolving cylinders, has been questioned by some metallurgists. In proof of its feasibility, Mr. Rothwell points out that, in the continuous arrangement above mentioned, ten tons of concentrates have been roasted in twenty-four hours, and to such perfection that in the subsequent chlorination by the Mears process, from 93 to 98 per cent. of the gold was extracted.

Condensation and Collection of Arsenic.—It has also been asserted by some metallurgists that the roasting of arsenical pyrites presents many difficulties, but experience with these Deloro ores has shown, on the contrary, that they roast with much greater facility, and in about two-thirds of the time necessary to roast simple sulphurets. They stand almost any amount of heat without fusing, and the arsenic, which forms 40 to 42 per cent of the mispickel, volatilizing at a comparatively low temperature, seems to leave the mass porous, thus facilitating the oxidation of the sulphur. The arsenic condenses readily in the series of brick chambers between the furnace and the stack.

The use of a centrifugal suction fan through which the furnace gases are drawn, and the draft of the furnace thereby created, is also believed to be a novelty in metallurgy, which at the Deloro has proved itself both practical and economical. It is clear that a blowing fan, which would occasion the escape of arsenical fumes, could not be used in this case.

Chlorination of the Roasted Concentrates.—The roasted concentrates are chlorinated by the Mears process, in charges of one ton, in a revolving lead-lined iron cylinder. The chlorine is made from chloride of lime and sulphuric acid, from 40 to 50 lbs. of the former and 50 to 60 of the latter being used to a ton of ore. The pressure in the cylinder rises to about 40 or 50 lbs. per square inch, and falls to 25 or 30 when the roast has not been so perfectly made as is desirable. The operation lasts about two hours, though probably less time than this will be found sufficient to completely chlorinate the gold when it is in fine particles.

Precipitation of the Gold.—The lime contained in the ore was found to give rise to quite unexpected difficulties in precipitating the gold from the chloride solution. The usual precipitant, ferrous sulphate, was found to throw down a voluminous precipitate of principally calcium sulphate along with the gold. In order to avoid this, an effort was made to get rid of the lime, first by sulphuric acid. This was too tedious, and, after many annoying delays, the precipitation by charcoal was tried. The chloride liquor is allowed to filter slowly through a mass of charcoal broken to, say, a sixteenth to half an inch. The barrels are kept full of solution by the filtrate being brought from the bottom of one barrel in a rubber tube which terminates a few inches below the top of the next barrel.

The precipitation of the gold is practically complete. The lime does not remain in the charcoal, yet the liquor undergoes such a chemical change by its contact with the charcoal, that the lime is no longer precipitated by either ferrous sulphate or sulphuric acid, but is by oxalate of ammonia.

Several chemists, who have experimented with this Deloro chloride liquor, have considered that hydrogen sulphide, either in a saturated aqueous solution or as gas, or ferrous chloride, would make convenient precipitants, either of these reagents precipitating the whole of the gold without the lime.

The charcoal which collects the gold is burned in an open pan with fire under, and (it is claimed) without loss of gold, at a cost said to be less than six cents per ton.

Chlorination of Concentrates containing Gold and Silver.—The roasting of concentrates of this description is carried out at the Deloro with the addition of salt, so as to convert the silver into a chloride. It is difficult to give any indications of how much salt should be used, and this has to be determined by experiment. It is not advisable to charge the salt with the ore into the furnace, but to give a desulphurizing roasting first and wait for the production of sulphates before adding the salt, which is then thoroughly stirred with the hot ore, and after thirty minutes to one hour withdrawn from the furnace on to the cooling floor, where it is piled up in a heap for as long as possible, as experience has shown that a beneficial result is obtained by leaving the ore in its red-hot state for some time, as no doubt the evolution of the chlorine gas, which is perceptible by the smell, continues to permeat the ore and thoroughly chloridizes the ore.

The ore after damping is charged into the lixiviation vats, and the gold extracted, according to the *modus operandi* of the Plattner process as previously described.

Operation for Extracting Silver.—After the gold has leached out and the wash water has drained out from the vat, a solution of calcium hyposulphite is allowed to flow into the same vat and by a separate launder is drawn into another precipitating tank.

The hyposulphite "leach," when holding chloride of silver in solution has a peculiar sweetish taste, and when this taste disappears it is an indication that the leach is finished, which is verified by test. Some of the leach flowing from the discharge-pipe is caught in a glass beaker and some solution of calcium sulphide is added. If a precipitate is produced it shows that the leach still carries silver or some other metal, and it is generally best to add some more hyposulphite.

The leaching is rather slow, and with some ores it takes two days, according to the character and silver contents of the ore under treatment.

Precipitating the Silver.—This is effected by means of a solution of calcium polysulphide, which throws down the silver and other metals as sulphides, the precipitate looking like dark mud. The addition of the polysulphide to the leach restores the hyposulphite into its original condition, so that the same can be used over again for leaching fresh charges; therefore this process can be considered as a continuous one. Care should be taken to avoid an excess of polysulphide, as its entrance into the leaching vats would cause a loss by reducing some of the dissolved chloride of silver into a silver sulphide, and enrich the tailings, and therefore in carrying out the precipitation it is better to stop the addition of polysulphide before all the silver is precipitated. Whatever silver remains in the solution will not be lost, as it passes through the ore again in the next operation.

A lively agitation of the liquid assists the precipitation, and when this is accomplished the renewed calcium hyposulphite is drawn off and pumped into the storage tank above. The silver sulphide precipitate is generally found to contain some base metal sulphides, and is drawn by means of a discharge faucet on to a filter, where it is washed with hot water, and by means of a press pressed into cakes which are slowly dried. The silver cakes are now roasted in a small reverberatory furnace, to drive off the sulphur, and care must be taken not to raise the heat too high so as to cause the melting of the cakes, which can easily take place if there is any lead in the ore.

The silver cake is now ready for melting, and this is accomplished in the usual way, by the addition of some scrap iron, which, along with some borax, is placed in a black-lead crucible and melted in the ordinary wind furnace. The iron is added for the purpose of combining with the sulphur to form a matte, and sufficient has to be added to combine with the sulphur. The matte collects on top of the molten metal and can be dipped off with a hot iron ladle, and when this is done some more silver cake, previously warmed, can be put into the crucible, and at the final casting of the ingot the matte collects on top of the silver ingot, and on cooling easily separates from it. If the matte is tough and not brittle, this indicates the presence of silver, and the same ought to be remelted, with the addition of more iron, which will generally yield a good button of silver.

The Adolph Thies Process.—This in reality is an outcome of the Mears process. The latter, while fairly effective from the metallurgical point of view, is objectionable in some particulars, especially the difficulty of keeping tight joints in the stuffing-boxes, and the consequent escape of chlorine gas through the works. The cost of repairs has also proved excessive.

The system adopted by Mr. Thies is extremely simple, and requires no tight joints liable to leakage. The chlorinator consists of a plain iron barrel lined with lead, and provided with a manhole on one side for charging and discharging the ore. The ore is charged in one-ton lots. The barrel is first partially filled with water, then a sufficient quantity of chloride of lime is dropped in, on top of which the roasted ore is charged. On top of the ore is poured the requisite amount of sulphuric acid, the manhole cover is put in place and thoroughly secured, and the barrel is started to revolve. The reason for charging in this order is to prevent the sulphuric acid and chloride of lime from coming in contact before the barrel has been securely closed. When the charging is completed the barrel is started and is kept revolving until the whole of the gold is dissolved. A very ingenious arrangement has been devised which enables the operator, by means of a clay pop-valve, to test from time to time, without opening the barrel, the presence of an excess of chlorine gas. Mr. Thies operated with some sulphides where each one-ton charge consumed 20 lbs. of chloride of lime and 25 lbs. of acid. The time occupied in chlorinating so far has been six hours, but with more experience this time ought to be reduced to one-half.

When this operation is completed, enough water is introduced to nearly fill the barrel, which is again revolved, so as to thoroughly wash the ore and dissolve the chloride of gold. The barrel is then opened and the liquor is decanted off upon large shallow filter beds. Another wash water is then put into the barrel, which is again revolved for a few minutes and then decanted upon the filter. Ultimately the whole charge is turned down in the ordinary way, and a final wash water is given on the filter.

The results so far obtained appear to have been remarkably successful. Subjoined are the results of eight charges, each of which contained a little over one ton of ore of such fineness that 99 per cent. of it would go through a 100-mesh and 60 per cent. through a 150-mesh screen. The assays made of the roasted ore showed $36·70, or £7 10s. gold per ton.

				$		s.	d.
Assay of tailings from	first charge gave		...	1·56	or	6	3
„ „	second	„	...	1·03	„	4	1½
„ „	third	„	...	0·78	„	3	2
„ „	fourth	„	...	0·52	„	2	1
„ „	fifth	„	...	trace.			
„ „	sixth	„	...	0·78	„	3	2
„ „	seventh	„	...	trace.			
„ „	eighth	„	...	0·78	„	3	2

In order to further test the advantages of this system, Mr. Thies tried one charge considerably heavier than the others, and endeavoured to filter it direct in the usual way. While he used for this charge nearly double the amount of wash water, and while the time occupied in filtering was much more than the other charges required, the tailings resulting contained $5·68, or £1 2s. 10d. per ton. There seems to be no difficulty in filtering through a bed of ore from 3 to 4½ inches thick; but when the thickness of the bed is greater, then the filtering of such very fine material becomes almost impossible, hence the advantage of decanting.

The precipitation is done in the usual manner with protosulphate of iron. If ores contain a considerable percentage of magnesia, it gives a voluminous precipitate. The precipitation of lime and magnesia may be entirely prevented if care is taken to keep the tank solution in proper condition. If such is not the case the magnesia shows in the precipitating tank a milky cloud.

The cost of roasting, labour, chemicals, and power, Mr. Thies states did not exceed in his operations $4, or 16s. od. per ton. The actual gold recovered is in all cases within a few cents per ton of the full assay value of the ore, less the amount shown by assay to have remained in the tailings. While the gold obtained in the stamp mill and amalgamating works is seldom purer than 897 fine, the gold obtained by this process is generally over 978 fine.

The Newbury-Vautin Process.—In this process also, chloride of lime is decomposed by sulphuric acid in a revolving barrel, and the evolved chlorine made to act on the gold in the barrel itself, rendering the manipulation very simple. The plant required is a hopper, chlorinating barrel, leaching vat or filter, reservoir for the liquid which is drawn from the filter, and the charcoal collector. The hopper is fixed above the chlorination barrel; into this the crushed ore, roasted pyrites, or tailings are poured.

The barrel is made of iron, lined with lead, this again with prepared wood; it is sufficiently strong to bear a pressure of 100 lbs. to the square inch. In form it is cylindrical, and in the centre are two valves directly opposite to each other. One valve is sufficiently large to allow the pouring in of a stream of tailings or sands; the other is smaller, and serves for the introduction of the compressed air. The barrels vary in size according to the quantity of ore required to be treated. The charge falls from the hopper through the larger valve into the chlorinating barrrel.

The chlorine is produced in the barrel by adding to the ore a certain quantity of chloride of lime and sulphuric acid, and enough water is added to produce a liquid mass. The valve cover is then screwed down, and the barrel turned half round, so that the small valve is uppermost. To this is attached an indiarubber pipe, leading from an air pump, and compressed air is forced in until a pressure of about 60 lbs. to the square inch is reached, when the valve is screwed down and the hose disconnected.

The barrel is now set revolving at a speed of about ten revolutions per minute, and the sulphuric acid decomposing the chloride of lime produces chlorine gas, which impregnates the water in the barrel, and the free gold is thereby converted into a chloride of gold, which is dissolved by the water. The time occupied to effect this solution ranges from one to four hours, according to the degree of fineness of the gold particles and the character of the ore.

The chlorinator, on being stopped, is now discharged first of the compressed air and gases, by means of the small valve, to which an indiarubber hose is attached leading outside the building. The large valve is then removed, and the barrel set revolving as before; at every downward turn it discharges a portion of its contents into a shoot leading to the leaching-vat below. When almost all its contents are discharged a few buckets of water are thrown in, which by the revolution of the vessel are swirled around, thus completely washing into the filter all remaining solution.

The filter is an iron vessel lined with lead, with a double bottom. To the lower part of this a pipe is connected leading to a vacuum pump. When the ore from the chlorinator has filled the filter, the air is drawn from between the double-bottoms of the latter, which causes very rapid filtration.

The filtered liquid is drawn off through a pipe into a vat;

a continuous stream of water is kept playing upon the tailings in the filter, which by the suction is drawn through them, so that all the chlorides are washed out. To prevent any clogging of the material, the action of the exhaust-pump is made intermittent. The solution as it runs to the vat being continually tested, it is easy to know when the work is completed; then the water is cut off and the pump stopped. The filtration generally occupies about one hour. The stuff in the filter is tipped into a truck running on a line of rails below, by which means it is run to the tailing heap.

The gold is now in the solution in the tank, at the bottom of which is a tap from which the liquid slowly runs through a filter of charcoal. Contact with this material returns the gold chloride to metallic gold, which collects on the charcoal. Copper, lime, magnesia, and zinc are not touched by this reagent, so they pass through in a soluble state. The charcoal after drying is burned, and the gold collected by it is melted into an ingot.

The Swedish (or Munktell) Chlorination Process.—In carrying out this process, Mr. Munktell prefers a chloridizing roasting. If the material after roasting and washing is found to contain gold in somewhat large grains, these may be conveniently separated by washing; or by some other mechanical means the heavy metals are removed.

He subjects the material, after being reduced to a sufficiently fine powder and freed from protoxides—whether roasted and separated from the coarse grains of gold or not, as circumstances may require—to the following process. The ore when pulverized should not be too fine, so as to allow liquids to percolate through it.

A small quantity of hypochlorite of lime, or some other salt in which the acid consists of chlorine combined with oxygen, is dissolved in water or in some other suitable liquid, such, for instance, as a solution of chloride of sodium, when the raw material contains silver. Instead of chloride of sodium, chloride of calcium or some other substance capable of dissolving chloride of silver may be employed. The solution should contain about one per cent. or less of hypochlorite of lime. This solution, before it enters the running tanks containing the auriferous material, meets with a highly diluted solution of some acid, such as muriatic acid, sulphuric acid, carbonic acid, or some other acid, the solution being of such a strength that equal volumes of the acid and the hypochlorite of lime will just suffice for the decomposition of the hypochlorite of lime.

The mixed liquids now percolate through the material, whereby the chlorine produced by the mixture, partly in a nascent state, unites itself with the gold, which is thereby dissolved, and is continually allowed to run off at such a rate that only a small excess of chlorine remains in the solution. This solution is led in to the precipitation tanks, where the gold, and any silver present, is precipitated by one of the usual methods.

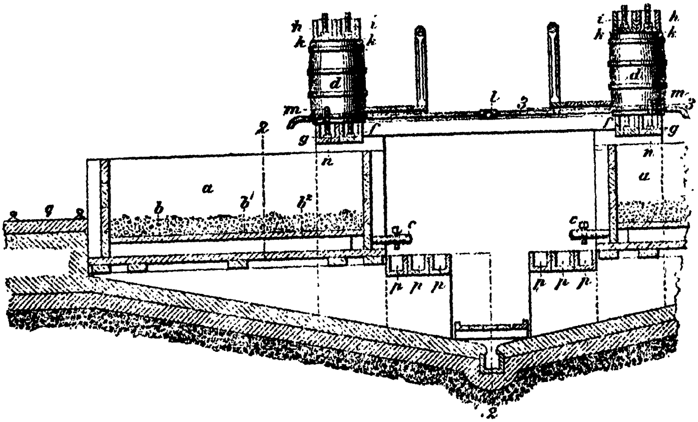

FIG. 101.—SECTION OF MUNKTELL'S PLANT.

This method of extracting gold can be carried out in a convenient and cheap manner without the escape of any considerable quantity of chlorine.

The solution may also be stronger and be mixed before hand, and then, after a longer or shorter time, be poured over the crushed raw material prepared in the manner described. In this case, however, not only does much chlorine escape, which has a very detrimental effect on the carrying out of the process, but also, larger quantities of the solution are required. By allowing the chlorine in a very diluted solution to act on the auriferous substances, and by producing this chlorine at the same moment the solution is used, the process becomes extremely convenient and inexpensive; and the difficulties (Mr. Munktell remarks) which hitherto have always been experienced when chlorine is used on a large scale for the extraction of gold are thus avoided.*

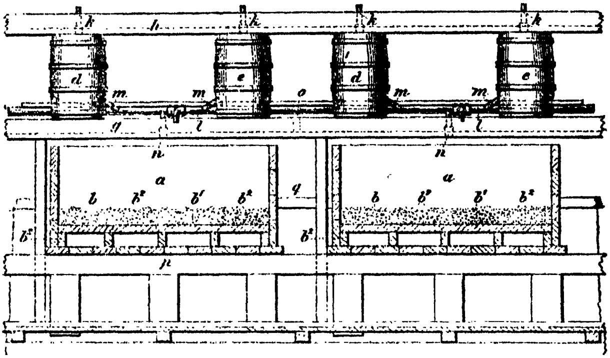

Fig. 102.—Longitudinal Section of Munktell's Plant.

The process can be carried out with very simple apparatus. A convenient arrangement consists of tanks having a layer of filtering medium above a false bottom. Over each tank are two vessels by which the quantities of the solutions are measured. They are filled by means of pipes or conduits and discharged into other conduits leading to the tanks. Other pipes or conduits carry water or other liquids to the tanks for dissolving and carrying away the protoxides, if any.

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* Mr. Munktell, however, when making this last statement, was evidently not aware of the inventions of Dr. Mears and others; and even with a properly arranged Plattner plant no inconvenience such as he anticipates need (I think) be apprehended.

Fig. 101 represents a simple arrangement of an extraction apparatus, showing the tanks in a section of the line 1—1 in Fig. 103. Fig. 102 is a longitudinal section thereof along the line 2—2 in Fig. 101; and Fig. 103 represents the same in plan and horizontal section along the line 3—3 in Fig. 101.

a a are extraction tanks of wood; *b b* represent filtering bottoms inside the tanks, consisting of a layer of gravel or quartz spread on perforated boards or plates of earthenware b^1, placed on the supports b^2. *c c* are outlet cocks. For measuring the requisite proportions of the solutions of hypochlorite of lime and the acid, the tubs *d d* and *e e* are placed on the channels *f f* and *g g* above the tanks. The diluted ready made solutions of hypochlorite of lime and acid are led separately through the channels *h h* and *i i* into the respective tubs

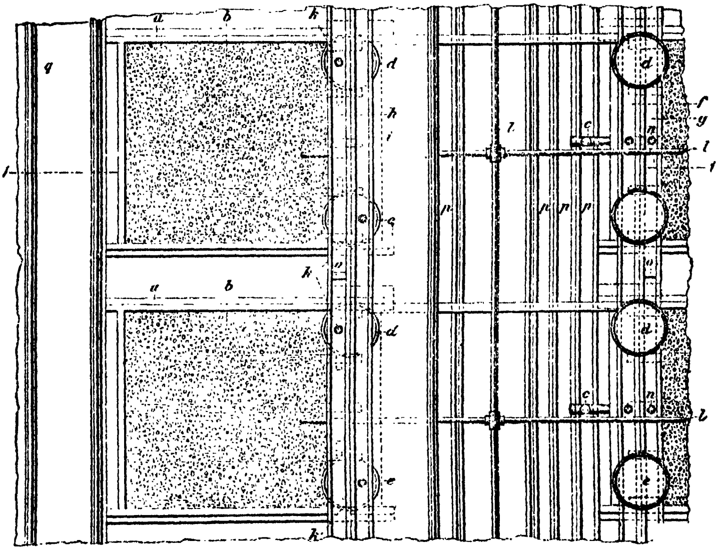

Fig. 103.—Plan of Munktell's Plant.

d and *e*. As each tub is filled, the plug, *k*, is inserted into the hole in the channel leading to that tub. The channels, *f f*, are used for admitting such acids and salt solutions as are required for dissolving other substances than gold. *l l* are water pipes.

After the tubs have been filled the process is commenced by removing the plugs, *m*, and the diluted solutions of hypochlorite of lime and acid are allowed to run in definite quantities out of the tubs, *d* and *e*, into the channels, *g*, meeting each other at the openings *n n* and thus running down together into the tanks. By cross boards, *o*, in the channels the tanks which are not to be operated upon can be shut off. *p*, *p*, *p* are channels to discharge the solutions which are running off or are let out from the tanks, *a*. By means of short shoots placed under the cocks, *c*, the liquid is led into the respective channels, *p*, and from there led to the precipitating tubs to be treated in the usual way; or the gold can be collected on a charcoal filter. Lines of rails, *q*, for transporting the material are also shown on the drawings.

The Ottokar-Hoffman Gold and Silver Chlorination Process.—If rich auriferous silver ores, in which the percentage of gold is high, almost equal to that in silver, should be subjected to a chloridizing roasting, then impregnated with chlorine gas leached with water, for the purpose of extracting the gold, and finally leached with a hyposulphite of lime for the silver, although a high percentage of silver might be extracted, the yield of gold would only amount to about 50 per cent., more or less.

This result is not easily explained. The gold may be influenced somehow during the roasting by the base metal chlorides preventing the gold from being attacked by the chlorine gas. On the other hand, if the base metal chlorides and the chloride of silver are extracted previously to the impregnation with chlorine, both metals, silver and gold, can be got out very close.

The *rationale* of the process is as follows: the concentrated sulphurets are subjected to a chloridizing roasting in a reverberatory furnace. These furnaces, although old-fashioned stirring furnaces, are considered preferable for this class of ores to any mechanical furnace in use, especially to the furnaces with continuous discharge. The concentrated sulphurets require a very perfect roasting, and Mr. Hoffman claims that the continuous discharge furnaces do not give time enough for very highly sulphuretted ore to become thoroughly desulphurized. Such high grade ores require close attention, and the process must be under perfect control by the roaster. However, it does not take nearly so much time as required by Plattner's gold chlorination.

After the ore has been roasted, it is spread on the cooling floor, and sifted when cool through a sieve of ten to fourteen meshes to the running inch. The sulphurets are heavy enough after roasting to make very little dust during the sifting, so that the inconvenient moistening can be obviated. The lumps are saved till a larger amount accumulates; they are then pulverized in a dry battery and slightly roasted.

The roasted and sifted ore is charged into tanks with filter bottoms, in quantities from 2½ to 3 tons, and leached with water to extract all soluble base chlorides. The water, if saturated with these base chlorides and some salt, of which some quantity is generally left in the ore undecomposed after roasting, acts on the chlorides of silver like a concentrated brine, and dissolves them. To prevent the escape of this dissolved part of silver, Hoffman does not admit the water from above the ore, as is usually done, but from the filter bottom, which, by means of a slight pressure, is forced to ascend through the ore to the top of the vat. In this way the concentrated solution accumulates above the ore, and in diluting it by a stream of water, and permitting the solution to flow out through the filter bottom, the chloride of silver is precipitated on and through the ore, which is then extracted, with the balance of the silver. The operation affects the fineness of the bullion somewhat if there is a considerable amount of lead in the ore, but not materially. The leaching of the silver is done as usual, by leaching the ore with a solution of hyposulphite of lime, and precipitating the silver with polysulphide of calcium.

After the silver has been extracted, the solution of the hyposulphite is allowed to run out till it disappears under the surface of the ore, when clear water is introduced again in order to displace all solution. The delivered ore is then removed from the tank to a dry kiln, where it is left for a time till the surplus water has evaporated. After this it is charged back into the tank, still moist. This second handling and drying cannot be avoided, as the ore after leaching is too wet to permit of a free passage of the chlorine gas; but if the works are arranged properly this partial drying causes neither much delay nor much expense.

The gold in the ore is now in metallic condition, and very bright and clean, permitting a very close extraction. The rim of the tank is provided with a groove, which is open toward the inside, 2½ inches deep and 1½ inches wide. In this groove fits the cover of the tank, leaving, however, a play of one-eighth of an inch around the circumference. The cover being made of inch boards, the staves of the tank will project 1½ inches above the cover. This arrangement facilitates the operation of making the cover air-tight. This is first done with clay, and then a sheet of water one inch deep is poured on the top of the cover, thus making it perfectly air-tight and preventing the escape of gas into the working-room, while the tank is charged with water for the purpose of extracting the gold. The cover is provided with two pieces of 1¼-inch gas-pipe, 6 inches long, and a square opening 6 by 6 inches. During the time of the impregnation of the ore with gas the pipes are closed with balls of clay. As soon as the charge is ready for the extraction of the gold, the balls are removed, and one of the pipes is connected with the hose of the water tank, while the other, by means of the hose, is connected either with another tank already prepared for chloridizing, or with the ash-pit of the roasting furnace. This is done to utilize the surplus of chlorine gas, and to protect the workmen from its very injurious effects. Care must be taken to have sacks placed on the top of the ore right under the water-pipe, and kept in place by weights—two bricks will do—to prevent the stream of water working into the ore. The square opening serves for examining the progress of the gas in the ore, and can be made air-tight by a good fitting cover and clay.

The chlorine gas is generated in a leaden gas generator, which is not heated as usual, by direct application of fire, but by steam. For this purpose the generator is placed in a tight-fitting box, leaving a space of two inches around the side and the bottom for the circulation of steam. The rim and cover are kept outside the box. The steam enters on one side through a half-inch pipe, while the other side of the box is provided with a one-inch exhaust pipe and an outlet for the water.

Whenever steam can be had in chlorination works, this arrangement will prove very convenient and useful. The operator has the temperature entirely under his control. The least turn on the valve increases or decreases the heat, and, of course, regulates the generation of steam. The discharge pipe of the generator projects a short way out of the box. On this pipe is fastened a piece of hose about 2 feet long, which can be closed with a thumbscrew clamp. The hose lies in a covered trough, which leads outside of the gas-house. In discharging the generator, a small stream of water is permitted to flow through the funnel into the generator, by which the gas is forced through its usual outlet into one of the tanks. When the generator is filled the stirrer is set in motion and the thumbscrew on the hose loosened. In this way the generator can be discharged without molesting or injuring the men.

The cover of the generator is closed tightly with clay, and only removed in case of repair. The gas is conducted through a leaden pipe, intersected with rubber hose. Each tank is connected with the main pipe, and can be disconnected by the use of the thumb-screw clamp. The pipe through which the gas enters the tank is independent of that through which the solution discharges. It is placed higher and as close to the false bottom as possible. After the gas has been a sufficient time in contact with the ores, the gold is extracted by water and precipitated in the usual way with sulphate of iron. The gold obtained is of extreme fineness, varying from 970 to 990 fine.

In treating very rich ores, containing, say, $700, or £140 to £180 gold per ton, the solution carrying out the gold is of a very lustrous yellow colour, and the precipitated gold accumulates on the bottom in spongy lumps of great specific gravity, some of them showing scales of bright gold, and some of these, as the microscope will show, may prove to be crystallized gold. In leaching very little more time is required for rich gold ore than for poor.

If the ore is copperous, considerable copper will be carried out with the gold solution, colouring it green. In order to save the copper, the solution, after the gold has been precipitated and settled, is decanted into the copper tanks; but before doing this it is advisable to draw the solution into a second gold tank, in order to catch the gold which may be accidentally carried off with the stream, and to keep it standing sufficiently long to allow the gold to settle again.

The Ottokar-Hoffman Furnace.—Mr. Hoffman has also invented an improvement on the ordinary revolving cylindrical furnaces, which can be used to advantage in oxidizing roasting of auriferous ores, as well as for chloridizing roasting of silver ores. In my work on the "Metallurgy of Silver" it will be seen that Mr. Aaron has used a furnace of this construction for roasting the Silver King ores. His improvement consists in combining with a revolving cylinder-furnace two fireplaces, one at each end, and connecting both with the furnace, and also with an escape flue. Suitable dampers are fitted between each fireplace and the escape flue, so that when a fire is built in one fireplace its damper may be closed and the flame directed through the furnace, and thence through the opposite fireplace and its flue to the chimney. After the ore at the end of the cylinder nearest the first fireplace is properly roasted and chloridized a fire is built in the other fireplace and the first one suffered to go down. The damper of the first one is opened, while that of the second one is closed, so that the direction of the heat is changed, and the ore nearest this second fire is subjected to the highest temperature. This enables the operator to roast and chloridize the ore quickly without exposing the ore at one end to an injurious temperature in order to heat that at the opposite end to a sufficient temperature. Fig. 104 is a longitudinal section of the fireplaces and flues.

The revolving roasting-furnace, as ordinarily constructed, consists of an iron cylinder, lined on the inside with bricks, and provided on one end with a fireplace, and on the other with a flue leading to the chimney. The shape of some of these revolving furnaces is prismatic, but in construction they do not differ materially from the cylinder-furnace. There are on the mantle of the cylinder four doors for charging and discharging. After the furnace is charged with ore and the doors are closed, the cylinder is set into a slow revolving motion, while the flames and gases pass in the direction from the fireplace to the flue, and through it into the chimney.

It is apparent that that part of the ore which is nearest to the fire will be exposed to a higher temperature than that on the farther end; and as the ore, by the revolving motion of the furnace, is mixed and turned, but does not change its respective place in the furnace, it is further apparent that no uniform roasting can be obtained, at least not without causing an unnecessary loss of silver by volatilization.

That part of the ore which is nearest to the fire is perfectly roasted and chloridized, while the ore at the other end is from five to ten and more per cent. less chloridized. In order to bring chlorination up, the temperature should be increased and roasting continued for a considerable time longer, which increased and continued heat disposes silver to volatilize. The same ore will lose less silver if the chloridizing roasting is completed in a short time than it will in a long period, and, again, will lose less if the chloridizing roasting is performed at a lower heat; but as these revolving furnaces are constructed (especially if long cylinders are used—say sixteen feet long), the metallurgist encounters great difficulties in treating the charge of ore in the furnace to the best advantage, and is not able to obtain sufficiently satisfactory results if it contains a great deal of antimony, and bakes easily, as the construction of the furnace does not give him the facilities to subject the ore to a uniform heat.

Antimonious ores have to be roasted at a very low heat; but if the fire is kept low enough for the ore nearest to it, the ore at the farther end of the furnace will not have heat enough to roast; and, again, if the fire is kept strong enough for the farther part, the ore nearest to the fire will commence to cake and sustain a heavy loss in silver.

To avoid these difficulties, and to enable the roaster to roast the ore at the required uniform temperature, an improvement in the construction of the revolving roasting-furnaces has been introduced by Mr. Hoffman. Opposite to the fireplace, and on the end of the cylinder where in the old construction the flue is, a second fireplace is erected, so that the furnace, A, is provided with two fireplaces, *b* and *c*, on each end one. Each fireplace is provided with a flue and damper, *d* and *a*, leading to the main flue, *e*, and chimney. It will be seen that by attaching the flue directly to the fireplace, and by shutting the ash-pit and fire-door and raising the damper in the flue during the working of the furnace, the firebox serves alternately as fireplace or flue, according to the way the flame is directed.

To bring the furnace into operation, after charging it with ore the fire is built in the fireplace, *c*, and the damper, *a*, closed. The flame and gases traverse the furnace, A, and pass through the opposite fireplace, *b*, and the flue into the main flue, *e*. After a lapse of an hour or more, according to the character of the ore, a fire is built in the opposite fireplace, *b*, the damper, *a*, opened, and *d* closed. Flame and gases are now forced to pass through the furnace in the opposite direction. This changing of the fire is kept up at regular intervals during the whole time of roasting. While one fire is in operation the other is allowed to go down.

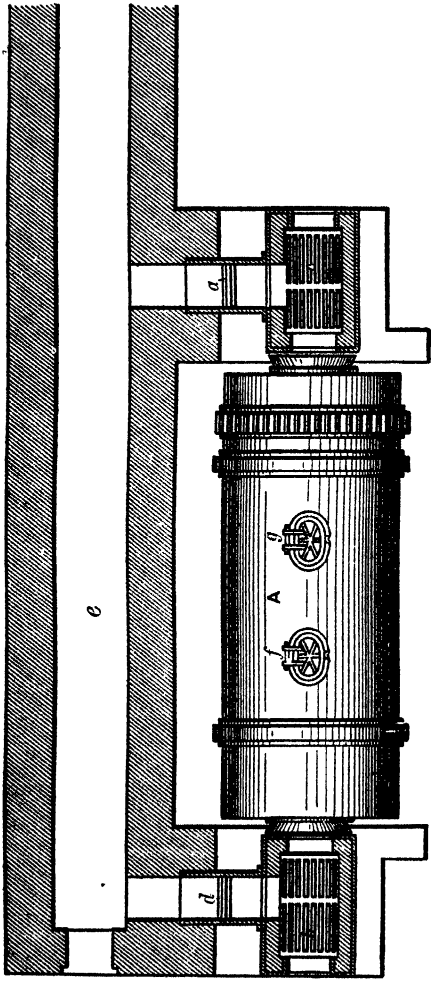

FIG. 104.—THE OTTOKAR-HOFFMAN FURNACE.

With these double fireplaces and flue arrangements, the ore in the furnace is subjected in both halves to the required roasting temperature, thus obtaining a uniform roasting throughout the whole furnace without the disadvantage of burning one half too much, in order to throw enough heat on the other. The construction of the furnace also permits the roasting of certain classes of ores, which bake easily in an ordinary revolving cylinder, and also prevents the dust from being carried by the draft into the flue during the charging of the furnace.

In charging ordinary furnaces the doors, *f* and *g*, are opened and placed under the outlets of a hopper. The ore falls in a shower into the furnace, and the draught passing through the furnace carries off considerable valuable dust into the flue. To avoid this, both dampers, *d* and *a*, are left open during the charging of the cylinder, and the draught passes through each fireplace and flue into the main flue independently of the furnace, through which no draught at all is passing.

There is one inconvenience with this furnace,—namely, that when the draught is carried over the fireplace filled with incandescent fuel, some ore dust must necessarily fall on to the fuel and cause clinkers, which have to be removed, and which hardly pay to treat, thereby causing a certain loss of metal.

The Von Pateras and Roeszner Process, which is based on the solubility of chloride of silver in a concentrated salt solution, is adapted for the extraction of gold, silver, and copper. The chloride of copper, like chloride of gold, is soluble in water, whereas chloride of silver is not, and would, without a salt solution, remain behind in the residue. If, therefore, chlorine is conducted through a salt solution to saturation, the chloridized salt solution will dissolve gold, silver, and copper simultaneously, and is accordingly adapted for ores containing these three metals.

Auriferous silver ores are therefore submitted to a chloridizing roasting, they are then charged into large vats and leached with the cold solution of salt saturated with chlorine. The leach waters are drawn off into the precipitating tanks and diluted with clear water, which produces a white precipitate of chloride of silver. The diluted salt leach has to stand now for twelve to fourteen hours before all the silver chloride has settled, and the fluid above it becomes clear enough to be drawn off into the gold-precipitating tank, where the gold is thrown down by a solution of sulphate of iron. When all the gold has settled, the fluid is drawn into the third copper-precipitating tank, which contains scrap iron, on which the copper precipitates.

This method of precipitation necessitates the subsequent reduction of the chloride of silver, by means of zinc and sulphuric acid, to metallic silver. The large quantity of water required to precipitate the chloride of silver—amounting to nearly fifty per cent. of the volume of the salt leach—dilutes the copper solution too much, and renders the precipitation of this metal difficult. It is better, by means of metallic copper, to precipitate first the silver and gold as metallic gold and silver, and then to draw off the liquid into another tank and precipitate the copper with scrap iron.

Roeszner roasts the ore with salt, extracts a part of the silver by Augustin's method with a hot solution of salt, and then treats the residue with a solution of salt and chlorine, and hot concentrated salt solution, alternately, for the extraction of the gold and the remainder of the silver.

According to Plattner, an auriferous and argentiferous substance, when roasted with common salt, forms chlorides of silver and gold. The chloride of gold is transformed into protochloride of gold, losing two equivalents of chlorine, when heated to somewhat below 200° C., and the protochloride of gold is decomposed into chlorine and metallic gold when heated to about 240° C. This easy decomposition of chloride of gold chiefly accounts for the imperfect yield of gold when treating silver ores by the amalgamation process, or submitting them to a lixiviation with common salt.

Tanikovitz published an account in the *Oesterreichische Zeitschriften* of 1863, in which he advised the submission of the ore first to a perfect oxidizing roasting, while admitting steam into the furnace, then the addition of salt, and chloridizing roasting at a lower temperature. Roeszner advises the leaching of this roasted mass with hyposulphites in preference to a solution of common salt and chlorine water.

Kiss has shown that roasting with salt produces a chloride of gold, which forms with hyposulphite of soda, or lime, a soluble double hyposulphite of gold and soda salt.

To prevent separation of metallic gold from the chloride of gold in the roasting process, the temperature must not be too high during the last roasting period, and some surplus chloride, produced from common salt and easily decomposable metallic sulphates, must be present: if none are in the ore, sulphate of copper should be added. It is advisable to wash the roasting mass with water previous to its treatment with hyposulphites. The chloride of gold, by the action of hot water, is decomposed into protochloride and metallic gold, and though insoluble in cold water is slowly decomposed by it in darkness.*

The distinction between Kiss's method and the Roeszner and Von Pateras method is that the former method is based on the fact that chlorides of silver and gold are soluble in a solution of hyposulphite of lime; and the process has been brought to great perfection in America—as far as the treatment of silver ores are concerned—by the improvement introduced recently by Mr. Russell, who substitutes hyposulphite of soda, with the addition of what he calls the extra solution. His process is described in my work on the "Metallurgy of Silver."

As before explained, in Von Pateras and Roeszner's process the ores or concentrations are submitted to a chloridizing roasting with salt, and the roasted mass is then treated with a solution of common salt saturated with chlorine. This process is modified by first leaching the ore, after a chloridizing roasting according to Augustin's method, with a hot concentrated solution of common salt; the residues are then alternately treated by cold solution of salt saturated with chlorine, and hot concentrated solution of salt, to extract the gold and the remaining silver. Both metals are precipitated from the hot solution by copper; the cement silver is cupelled to drive off all impurities, and the gold separated by parting with acids.

Electro-Chlorination of Gold Ores: Cassel's Process.—Many attempts have made to obtain the precious metals by means of electrolysis, and several ingenious methods have been devised, but so far no practical success has crowned these efforts, although the results hitherto obtained on a small scale warrant the expectation that eventually the difficulties which still preclude the working of these processes on a large scale will sooner or later be overcome.

A Glasgow company is developing the Cassel process, which is one of chlorination. The apparatus consists of a large drum, within which are arranged a number of dense carbon rods; these rods form the anodes, or positive electrodes, and are metallically connected with the positive pole of the dynamo, while the negative pole of the dynamo is connected with the hollow iron shaft of the drum, which serves both as axis to the drum and also as negative electrode of the apparatus. This hollow shaft terminates through stuffing boxes in hollow standards or tanks, where finally the gold accumulates.

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* See Crookes and Röhrig's "Practical Treatise on Metallurgy."

In carrying out the processes, the drum is charged with 4,000 to 5,000 lbs. of ore, and salt and water are added. This drum is set revolving slowly—eight to ten revolutions per minute. The electric current being turned on, the chloride of sodium decomposes, and chlorine and oxygen are generated at the anode. During the revolution of the drum the ore comes into contact with the carbon elements, which discharge the chlorine from the salt solution. The gold particles thus acted upon are converted into terchloride of gold, which, as soon as formed, dissolves in the solution.

Mr. Cassel, it seems, had to contend with the difficulty that hydrochloric acid was formed during the reaction. This dissolved the iron oxide present in nearly all ores, roasted iron pyrites being the materials generally operated upon. That difficulty he overcame by adding caustic lime, which neutralized the hydrochloric acid as fast as formed, so that no iron could be taken up by the gold solution. As hypochlorite of lime is formed, this again is decomposed by the water present, and gives up its chlorine to be taken up by the gold.

The ultimate products of the reaction are chloride of sodium, which has not been all decomposed, chloride of calcium, terchloride of gold with the gangue at the anode; and chloride of sodium and caustic soda at the cathode. In the iron shaft are bored a number of holes, and the shaft itself is covered with asbestos cloth, which, while preventing the gangue from entering the shaft, allows the dissolved gold to penetrate through the cloth. After the addition of the lime, which precipitates all other dissolved metals present except the gold, the latter metal is rapidly dissolved, and is deposited by the electrical action in the interior of the pipe in a finely divided metallic state; thence it is carried into the hollow standards by means of an Archimedean screw fixed in the pipe. The chloride of gold is decomposed at the cathode into metallic gold and chlorine. The standards are provided with movable doors, from which the gold precipitate is withdrawn, and after drying is melted into ingots.

From this description it will be seen that chlorine can be produced by electrolysis from salt solutions; and although the above method does not seem the most practical way of applying it for the production of the terchloride of gold, it is likely that improvements will follow which will perfect Mr. Cassel's otherwise excellent idea. Gold is more readily attacked by nascent chlorine gas than by chlorine held in solution by water; and it seems that pressure also assists rapid chlorination, as is shown by the discovery made by Mr. Mears, as well as by the Newbury-Vautin process.

Hauch's Treatment of Telluride Gold Ores.—A method of treating these ores which has been devised by Anton Hauch, Assayer to the Hungarian Government, deserves notice here. The gold ores of Nagyag and Offenbanya are highly prized on account of their contents in tellurium, which is rarely met with in mines. Tellurium is important on account of its employment in the manufacture of thermo-electric batteries, and an expeditious and cheap method for its extraction from ores is, therefore, much to be desired. The analysis of the Hungarian ores gave the following results: tellurium, gold, silver, a considerable percentage of quartz, carbonate of lime, and carbonate and sulphide of manganese.

The idea which the inventor carried out was to produce the tellurium by means of a cheap acid solution, from which it could be easily separated. In defining his method, the manganese combinations had to be closely studied. A quantitative determination of the ores gave the following results: 30 to 40 per cent. of quartz; 10 to 20 per cent. of carbonate of lime; 15 to 20 per cent. of carbonate and sulphide of manganese; 5 to 8 per cent. of lead sulphide; 1 to 2½ per cent. of copper sulphide; 5 to 8 per cent. of clay; 1 to 4 per cent. of zinc sulphide; with some cobalt, nickel, antimonium, and arsenic, besides tellurium, gold, and silver. If the ore was submitted to an oxidizing roasting, a portion of the tellurium volatilized, carrying off some gold and silver, which loss could be avoided by condensation.

During the roasting the carbonate of manganese and any manganese sulphide present were converted into manganese oxide, the sulphide of manganese being no doubt influenced by the presence of lime. Under the influence of hydrochloric acid the manganese oxide gave chlorine. It was found also that during roasting a large portion of the gold became metallic, in such a way that fifty per cent. of the gold could be extracted by amalgamation.

In treating the roasted ores with dilute hydrochloric acid in lead-lined wooden tubs provided with a stirring apparatus, a disengagement of chlorine takes place, which converts the metals to be obtained into a soluble condition, with the exception of silver. The surplus chlorine is conducted into tubs filled with water, and this chlorine water can be used as a solvent for the tellurium sponge. After the solution is drawn off from the ore, sulphuric acid is added, which precipitates gypsum and lead sulphate.

A further decantation of the solution from the precipitate is effected, and a solution of iron sulphate is added, which precipitates the gold. The solution is now filtered off, and further treated with metallic zinc, which precipitates the tellurium as black sponge. This tellurium sponge is washed with water acidulated with hydrochloric acid, filtered rapidly, dried and melted, without any further addition, in a platinum crucible, giving raw tellurium, which always contains some lead, copper, nickel, and antimonium.

If the tellurium sponge is first dissolved in chlorine water, then treated for a considerable length of time with sulphuric acid, pure tellurium is obtained and can be melted down as such. After the separation of the largest proportion of the gold and the tellurium, there remains chloride of silver and some gold in a still soluble condition in the solid residues. In treating the damp residues with a solution of iron sulphate, the gold is converted without loss into its metallic state. From the residues, both silver and gold can be obtained by amalgamation, and where practicable can be smelted down with lead.

A practical trial on 7 kilos of telluride ores, containing 21·25 grams gold and 21·06 grams silver, gave the following result:—

The oxidising roasting in a muffle furnace took an hour and a half and the roasted ore weighed 6·498 kilos; there was, therefore, a loss in roasting amounting to 0·502 kilos, or 7·2 per cent. The contents of the roasted ore were 21·175 grams gold, and 20·25 grams silver; therefore the loss in roasting in gold 0·075 grams, or 0·35 per cent., in silver 0·81 grams, or 3·87 per cent. From these 6·498 kilos of roasted ore only 6 kilos were taken for the operation, containing 19·55 grams gold and 18·70 grams silver. These 6 kilos were introduced slowly and under constant agitation into a mixture of 3 litres water, 2 litres raw hydrochloric acid (20° B.), and 0·3 kilos concentrated sulphuric acid. An active development of carbonic acid and disengagement of chlorine took place, and the mass foamed up considerably. After twenty-four hours, during which time the mixture is repeatedly stirred, 2 litres of water were added, and then left standing quietly for two hours, and the solution, which was not as yet quite clear, was then decanted. The solution was renewed three times and the resulting 10½ litres of brine were treated with 2 litres of iron sulphate solution (25° B.) and well stirred. The metallic gold was completely precipitated after twenty-four hours, and the liquid separated by decantation. The residue after filtering was dried and melted, with an addition of lead; and after cupellation 16·67 grams, or 82·2 per cent. of the pure gold was obtained. This product could have been increased to 90 per cent. if a fourth solution had been used.

The brine decanted from the gold was treated with 2 kilos metallic zinc. The tellurium precipitated after twenty-four hours as a black sponge. After decantation, filtering of the residue, drying and smelting, 30 grams of raw tellurium were obtained, or 0·43 per cent. of the weight of the ore. The richness of the ore in tellurium was found to increase with its contents in gold.

For the precipitation of the tellurium 200 grams zinc were required—that is, 3 per cent. of the weight of ore. The residue, containing some iron sulphate solution, weighed 5·25 kilos; consequently the loss in lixiviation was 0·75 kilos, or 12½ per cent. This residue contained 3·88 grams gold and 17 grams silver. From the solution was obtained 16·07 grams gold, and adding 3·88 grams contained in the residue, makes 19·95 grams gold, which according to the assay contents of 19·55 grams proves a gain of 0·40 grams, or 2 per cent.

The assay showed 18·70 grams silver, and the residue contained 17·03 grams; there was a loss, therefore, of 1·67 grams, or 8·9 per cent. This discrepancy can only be accounted for by the impossibility of obtaining from the ores a perfect assay sample to represent a correct average; and, further, during the assaying process, either in smelting of cupellation, the volatilization of tellurium will carry off some or the precious metal with it.

This method can be carried out on a large working scale, and will prove a cheap mode of extraction for the metals contained in tellurides.

CHAPTER VIII.

SMELTING OF GOLD ORES AND REFINING OF GOLD BULLION.

THE HUNGARIAN METHOD OF SMELTING WITH IRON PYRITES—Cupellation—As practised in London—Cupelling Furnaces on the Continent—The Parting Process—Parting with Sulphuric Acid—Mint Parting with Nitric Acid—Refining of Brittle Gold—Miller's Process for Refining Gold with Chlorine Gas.

The Hungarian Method of Smelting with Iron Pyrites.—I think I may claim for Hungary—my native country—the introduction of a system by which gold is extracted from ores *by direct smelting*, a method which is practised there with iron pyrites containing a small quantity of gold.

When this mineral is heated to a certain temperature without access of air, it loses half its sulphur, and becomes converted into a fusible sulphide of iron. In treating auriferous iron pyrites contained in a quartz matrix, a flux must be added to melt the quartz; a slag will then be formed, and the sulphide of iron will unite with the greater portion of the gold present, forming a heavy regulus, which will subside to the bottom. The stratum of sulphide of iron thus formed beneath the surface of the slag and scoriæ produced by the fusion of the earthy and silicious gangue is technically termed a "matt." If this matt, containing the gold, be now roasted, or oxidized by heating it with free access of air, whereby it is deprived of a further portion of its sulphur, and the product be mixed with a fresh quantity of crude auriferous pyrites, and smelted again, a second matt will be obtained, which will contain the gold from the first and second charges of the auriferous ore. This process may be repeated a third, fourth, or any number of times, until the precious metal has been sufficiently concentrated in the substratum of sulphide of iron to admit of its profitable extraction. The last matt, containing the whole of the gold which has accumulated from the successive charges, and brought, therefore, to any required richness, is roasted and then fused with metallic lead, litharge, or with a mixture of galena and metallic iron. The gold is abandoned to the liberated lead, from the top of which the impoverished undecomposed sulphide and oxides may be skimmed off.

By this process the gold is obtained in a more concentrated form, or mixed with a smaller proportion of lead, than when the latter is used directly as a means of separating and collecting it from the smelted ore. The employment of the iron pyrites in the manner above described is only attended with advantage when it constitutes part of the ore itself, or when it can be readily and cheaply procured in sufficient abundance. When it forms a principal ingredient in the ore, it is manifestly highly advantageous, in point of economy, to employ it as a vehicle for the concentration of the precious metal, while it contributes at the same time to increase the amount of the product.

I know only of one locality in the United States, and that is in Montana, where this process has been put into practice and has proved a success.

Cupellation.—Another important metallurgical operation is the extraction of the gold from the lead, and this has been performed from the most remote antiquity by a process called cupellation. This process, which is very simple in principle, is at the same time one of the most elegant metallurgical operations ever invented. It depends on the circumstance that when silver and gold are exposed in a state of fusion to the action of the air or oxygen, they neither give off perceptible vapours nor are sensibly oxidized, they remain, in short, unaffected; whereas, under similar circumstances, lead and almost all the other metals are oxidized with greater or less rapidity. There is also an important peculiarity connected with lead, which renders it the only metal, except bismuth, applicable to the process of cupellation. This peculiarity consists in the circumstance that the oxide of lead, known as litharge, is fusible at a bright red heat, and in this state absorbs any other metallic oxide with which it may happen to be in contact, but which, without the influence of the oxide of lead, would remain uninfluenced by the heat. For example, if copper be present as an oxide it is taken up by the melted oxide of lead; and if the latter can be separated from the gold and silver, the oxide of copper will be expelled along with it.

This separation may be effected in various ways, but particularly by means of bone-ash, which possesses the peculiar, property of absorbing the melted litharge even when mixed with a certain proportion of other oxides, whereas it remains impermeable to the unoxidizable metals. Hence, if a vessel be formed of pounded and compressed bone-ash, and into this vessel a mass be introduced, consisting of a mixture of litharge or oxide of lead, copper or oxide of copper, and native gold (which always contains a certain proportion of silver), and if this mixture be exposed to a considerable heat, the oxide of lead will dissolve the oxide of copper, and both will be absorbed by the bone ash, while the mixture of gold and silver will remain in the vessel, melted, but otherwise unaffected. Hence it follows that gold or silver, or a mixture of the two, may readily be deprived of copper or any other oxidizable metals by cupelling with oxide of lead. For this purpose it is not necessary to employ the oxide itself in the first instance. When metallic lead is heated to a temperature above its melting point, it rapidly combines with the oxygen of the air, the product being in the first place yellow oxide of lead, which appears as a scum on the surface; and when the temperature is raised to about bright redness, this oxide is converted into litharge, or the fusible oxide, which immediately melts. Pure copper requires no less than sixteen or seventeen times its weight of lead to absorb it in the manner above mentioned.

In carrying out the cupellation process on a large scale we do not utilize the absorption power of the cupel alone, but are assisted by another agency, which is compressed air.

The English method of Cupellation or refining, as carried out in a large refinery located in the heart of the city of London,* is as follows:

In preparing the cupel, a piece of flat wrought iron, about ⅜ to ½ inch thick and 4 to 5 inches wide, is bent into the form of an oval hoop and the ends are welded together. The larger diameter of the hoop is in conformity with the size of the cupelling furnace and varies from 4 to 5 feet, and the short diameter from 2½ to 3 feet. Five or six flat bars are fixed to the lower part of the hoop, arranged parallel to each other in the direction of the short diameter. The first bar is placed about 9 inches from one end of the oval, and the others at equal distances between this bar and the other extremity. The hoop is then placed with the cross bars downwards upon a solid floor, and a quantity of pounded and sifted bone-ash is beaten firmly into it with a wooden rammer until it is entirely filled, or the bone-ash is level with the upper edge of the hoop. The bone-ash employed for this purpose is mixed with about one-tenth, by measure, of fern ashes, or one-fortieth by weight of American pearl-ashes, and moistened sufficiently to become coherent by pressure. The fern or pearl ash has the property of giving greater consistency to the bone ash when heated. When the hoop or frame has been well filled with this mixture and solidly beaten down, the surface is carefully scooped out with a trowel all round the centre, so as to form a shallow concavity of about 2¾ inches in depth, leaving a wall of bone-ash all round, about 2 inches in thickness at the top, and 3 inches at the bottom, except at one end, where a thickness of 13 inches is left; whilst the thickness of the sole itself is reduced to one inch above the upper surface of the iron cross pieces. At the end of the test where the wall is left 13 inches thick—which is termed the breast—a segment of the bone-ash contiguous to the hoop is removed.

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* The refinery of Messrs. Browne & Wingrove.

This apparatus is represented in Figs. 105 and 106, of which the former is the plan, and the latter a longitudinal vertical section. In these drawings the letters A A denote the iron hoop; *a a* the transverse bars; E indicates the cavity in the bone-ash; B, the breast; and F, the space where the bone-ash is removed between the breast and the iron. In cupels of more modern construction the bone-ash is not removed from the part F, as the litharge is found to come in contact with the iron ring, on which it exerts a powerfully corroding action, when such an arrangement is adopted. In place of this, however, a round hole is cut in the centre of the breast near F, to which channels are cut.

When fully prepared, the test is allowed to dry, and is then placed in a furnace, constructed in all respects like a common reverberatory furnace, except that a space is left open in the bed of it to receive the test, and that the width of the arch is much reduced (see Figs. 107 and 108). The test, when placed in position, forms the bed of the furnace, with the long diameter transversely. The test is placed on a small flat bogie car *c* built of iron, whose platform can be raised by means of three jack screws *g*, and the part F of the test projects beyond the platform of the bogie. The whole is run now under the arch forming the lower part of the furnace, and by means of the jack screws the test is raised into position, thus forming the hearth of the furnace. The fireplace *a* of a furnace is

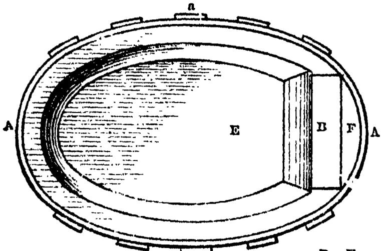

Fig. 105.

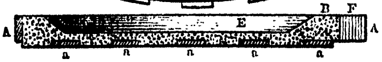

Fig. 106.—Cupel.

generally 18 inches wide by 2 feet 4 inches long, or less, and is separated from the body or bed—and in this case, the cupel—by a fire bridge *b* 18 inches wide, which allows the flame to pass directly over the surface of the cupel H, from whence it passes into a chimney *p*. At the side of the furnace, opposite the breast of the cupel, is an opening W through which passes the nozzle of a continuous pair of bellows.

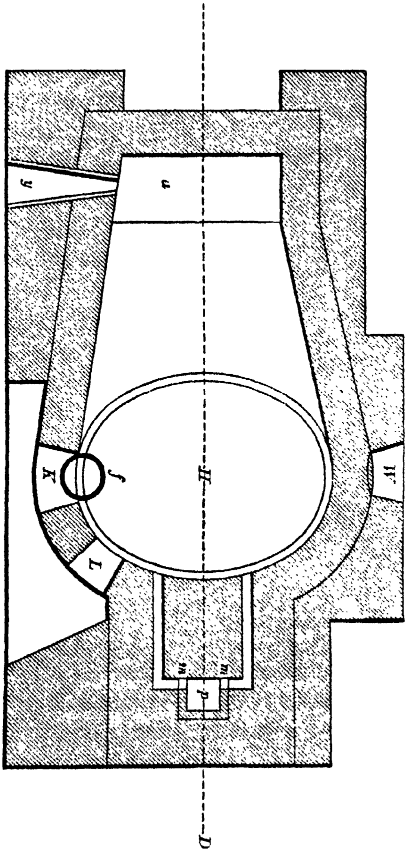

FIG. 107.—PLAN OF CUPELLATION FURNACE.

When introducing a fresh cupel or test, the fire must be low and heat must be applied with great caution, or otherwise the bone-ash will split to pieces; and for the same reason the bone-ash must be dried very gently.

When *silver lead* containing gold is to be cupelled, it is first smelted in a cast-iron pot set in a furnace near at hand, and after the cupel is red hot, the metal is ladled in till the cupel is nearly full. The blast is turned on, and oxidation sets in very quickly. At first the lead becomes covered on the surface with a yellow or grayish dross, but on further raising the heat, the surface of the bath "uncovers," and a film of melted litharge begins to appear. Part of this litharge sinks into the bone-ash of the test, which it soon saturates. The blowing apparatus at the back of the test is now set in motion, and forces the litharge as it forms forward to the breast, B, across the surface of which a channel is cut, through which the litharge begins to flow, and runs over at the end through the vacant space, F, into a movable iron pot, *f*, placed on the floor

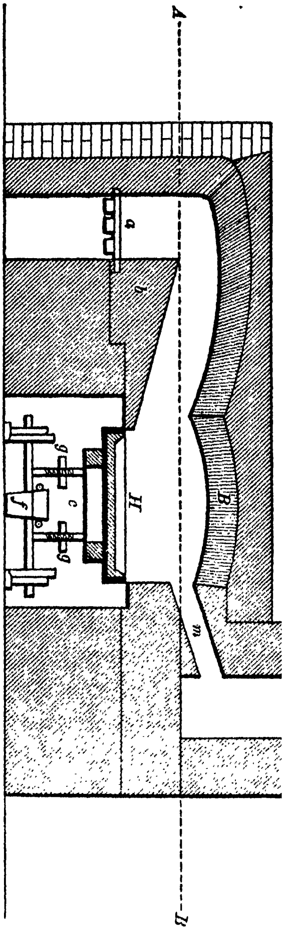

FIG. 108.—SECTION OF CUPELLATION FURNACE.

for its reception. The current of air introduced at the back of the cupel not only assists in clearing away the litharge, but supplies the amount of oxygen necessary for its rapid formation. In proportion as the litharge flows away, more of the melted lead is gradually supplied to the test, which is done by ladling it into a channel from the pot outside the furnace through the opening L. After a certain period, the channel first formed in the breast, B, becomes much corroded; it must then be stopped and a second and third channel successively formed for the same purpose. In this manner the operation may be continued until six or eight tons of the rich lead have been refined on the same test.

In dealing with doré silver containing a large percentage of copper, the refining is carried out by first melting the doré silver bullion in a wind furnace in crucibles, and ladling it into the previously heated cupel and adding molten lead in sufficient proportion to carry all the copper away. The proportions of lead necessary to purify the silver are substantially identical with those given in the chapter on assaying of bullion (see page 268).

Towards the conclusion of the operation some striking appearances are often presented. If the gold present is mixed with a considerable amount of silver, the surface of the mass, as it cools, is suddenly thrown into agitation; cones, or little craters, sometimes several inches in height, are thrown up, from which oxygen gas escapes with violence, projecting particles of silver with considerable force. This phenomenon is termed "spitting," or sometimes "vegetation," from the beautiful arborescent forms which are presented, and arises from the circumstance that metallic silver, when melted, has the property of absorbing six times its volume of oxygen, which is ejected with violence at the moment of solidification.

After cupellation, if it is not desired to solidify the metal in the test, the latter can be lowered by means of the jack screws; the bogie is run out from under the furnace arch, and the metal ladled quickly into moulds; or a mould can be placed under the test and a hole punched in the bottom, so as to allow the metal to flow in a regular stream into the mould.

On the Continent, the process of refining is conducted without the use of a cupel, properly so called, no bone-ash being employed. The test, if it may be so termed, is the bed of the furnace itself, which consists of a kind of marl, firmly beaten down into a circular cavity, which slopes from the sides to the centre and is allowed to dry. The roof of the furnace, which consists of a flat dome of bricks, built in a strong circular hoop of bar iron, is movable, by means of a crane. Several tons of lead are introduced at one charging, and after the roof has been replaced, the blast is transmitted through one or more apertures in one side of the furnace. The whole mass of metal is then worked off continuously, without any addition, till the lead is removed, and the alloy of gold and silver is left approximately fine.

The Parting Process.—The final operation of separating the silver from the gold is termed "parting," and is generally performed in England, in private refineries, by means of nitric acid, which dissolves the silver without attacking the gold. On the Continent and in the United States refineries the same effect is produced with hot sulphuric acid.

Parting with Sulphuric Acid.—The operation is very simple, and if any copper has been left in the alloy, it is separated along with the silver. Its presence is by no means injurious. In this process the proportion of gold must be less than in the parting with nitric acid, and the proportion of the gold to the silver should not exceed one-fifth. Experience has shown that the alloy should not contain more than one-twentieth part of copper. When the silver is not already present in the required proportion—which is never the case with native gold—the requisite excess must be introduced; for when the amount of gold exceeds the proportions above stated, the particles of silver are so enveloped in that metal as to resist for
time the action of the strongest nitric or sulphuric acid.

The alloy being prepared in the requisite proportio
is generally done by introducing the excess of silver before

cupellation—it is melted in a crucible, and granulated by pouring it into cold water. For one part of the granulated alloy three and a half parts of concentrated sulphuric acid are taken, and the mixture is put into a platinum vessel, which is then introduced into the furnace. The vessel should not be more than two-thirds filled, to guard against the effects of effervescence, which might throw out part of the liquid. It is then covered with a platinum hood, provided with a beak or tube for conveying the gases and vapours into a condensing apparatus. The capacity of the platinum retorts varies. After two or three hours' boiling, varying the time according to the size of the retorts, the silver and any copper that may be present are completely dissolved.

Owing to the great expense of platinum retorts, Mr. Tocchi introduced iron retorts, which can be safely employed, although the former are in general use. He found that the very concentration of the acid presents, of itself, an obstacle to the precipitation of the silver or copper by the iron; and, further, that the interior surface of the retort acquires a coating of silver, so that the iron and the liquid soon cease to be in contact. This point has not been sufficiently studied, but the fact has been placed beyond doubt that the operation succeeds very well in iron vessels, and accordingly, in many refining establishments, no others are employed.

It cannot fail to be observed that, assuming the proportions above given, the quantity of sulphuric acid is much greater than would be strictly necessary to convert the silver and copper into sulphates. Supposing the alloy to be as poor as possible in gold, theory would indicate the following quantities of acid as being sufficient for this purpose:

	Parts.		Parts.	
Copper	50	would require	155	sulphuric acid.
Silver	949	„ „	861	„ „
Gold	1	„ „	0	„ „
Alloy	1000		1016	

But as the amount of sulphuric acid employed is actually

3,500 to 1,000 of alloy, there is an excess of about 2,500 of acid, intended to hold the sulphates, especially the sulphate of copper, in solution.

When the silver and copper are completely dissolved, the retorts are withdrawn from the fire, and the solution is left to cool, in order that the gold may settle to the bottom. The liquid is then decanted; the gold, which remains in the vessel, is carefully washed, and the water employed in this operation is added to the sulphate solution. Lastly, the purified gold is melted, and after being cast into ingots, is ready for commerce.

The acid liquor containing the sulphates is poured into a leaden boiler containing water and copper shavings. Heat is applied and the sulphate of silver is soon completely decomposed. The precipitated silver is collected and submitted to repeated washings, always adding the water to the contents of the boiler, to be treated in the next operation. Lastly the silver is dried in a small iron pan, and is then melted in a crucible to be cast into ingots. The acid solution, which now contains only sulphate of copper, is evaporated in the lead boilers till it is fit to crystallize. It is then put into the crystallizing pans, and the mother waters are further evaporated to obtain more crystals. This process is continued till the liquid becomes very concentrated, and consists almost entirely of sulphuric acid. In this state it is termed "black acid," on account of its colour, which is due, in great part, to organic matters, dust, and other impurities, which fall in it during the evaporation in the crystallizing pans. The black acids may either be employed to repeat the same operation, or turned to useful account in different manufacturing processes, which do not require that the sulphuric acid be pure if highly concentrated.

Mint Parting with Nitric Acid.—In the next chapter, on Assaying, I explain that when gold has to be separated from silver, the last named metal must be present in the alloy in the proportion of at least 2½ to 1, otherwise the gold will so cover the silver as not to permit a perfect separation of the metals even if boiled for some time in nitric acid. The usual proportion in practice is 3 silver to 1 gold. Therefore, if we have a gold alloy weighing 1,000 ounces, 800 fine in gold, this will require—supposing the other 200 parts to be silver—$(3 \times 800) - 200 = 2{,}200$ ounces of fine silver, which has to be melted together with the 1,000 ounces of alloy. After fusion, the molten mass is granulated by being poured into a large vat filled with cold water. The object of this granulation is to reduce the metal into small particles of various shapes, offering a large surface to the acid, which quickens the operation of dissolving.

The silver granules can be dissolved in glass retorts, with openings leading to a high chimney, so as to draw off the copious nitrous acid fumes. The retorts are placed on a sand bath. The gold remains behind as a brown powder, which after the nitrate of silver solution is drawn off into large stoneware tanks, is washed, dried and melted in Morgan's black-lead crucibles with borax and saltpetre.

To the nitrate of silver solution is added a concentrated solution of common salt—chloride of sodium—which throws down the silver as a white curdy precipitate of chloride of silver. After drawing off the acid this chloride of silver is now reduced, by the addition of metallic zinc and water acidulated with sulphuric acid, to metallic silver, which is very light and spongy. It is collected in filters, washed for some time so as to free it from any soluble salts, and compressed under a powerful hydraulic press into cylindrical slabs, which are dried in ovens. After drying they are heated and assume a metallic lustre, and are now ready for the melting-pot. This operation gives silver nearly 1000 fine.

For making it into coin 900 parts of this silver, after being cast into ingots, are melted with 100 parts of refined copper, which makes the standard American alloy for the silver coin.

Refining of Brittle Gold by Chlorine Gas.—Mr. F. B. Miller, Assayer in the Sydney branch of the Royal Mint, has ed a method of refining gold which is perfectly suitable for removing such metals as lead, arsenic, antimony, and bismuth. These alloys, if only forming the $\frac{1}{1920}$th part of the whole mass, render the metal brittle, and totally unfit for coinage and the arts. The process consists in passing chlorine gas through the metal, while in a molten state, for about four or five minutes. The base metals are thus converted into volatile chlorides which escape; and should there be any silver present it is converted into chloride of silver which floats on the surface of the metal.

In order to avoid loss of silver, an earthenware crucible should be employed which has been previously dipped in a solution of borax; borax should also be introduced into the crucible with the gold.

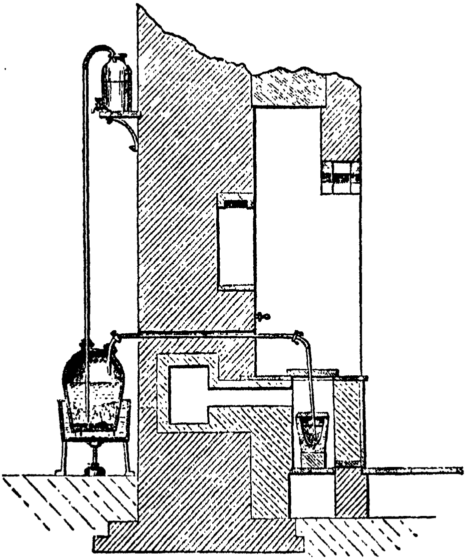

FIG. 109.—REFINING APPARATUS.

Fig. 109 shows the arrangement as adopted in the Royal Mint, in London, by Mr. Roberts-Austen (Principal Assayer), for refining brittle gold by Miller's process, operations for the purpose having been conducted there on an extensive scale. The chlorine is generated in an iron vessel having a capacity of 8 gals., and situated behind the flue. From it the gas is led by means of a long clay pipe underneath the surface of the molten metal, the pipe passing through the cover of the crucible, which, of course, should not be luted.

Mr. Miller has described his process as follows:—

"There is no recorded instance of gold having been found in an absolutely pure state. Every natural alloy of native gold contains more or less silver; and in almost all bullion resulting from the melting of alluvial gold the portion that is not gold consists chiefly of silver, with only a very small portion of foreign metals, usually copper and iron, with occasionally a little lead or antimony, and sometimes a trace of tin, iridium, &c. This, however, though true generally, is not always the case with gold obtained from quartz veins by amalgamation, as the mercury occasionally reduces and takes up other metals as well as the gold, which appear in the bullion on melting. Where silver is associated in varying proportions with the gold the question will naturally arise, Is this argentiferous character in any way connected with the geological structure of the district? It is a fact, and certainly a very curious one, whether it arises from accidental causes, or whether it may hereafter be traced to peculiarity in the rocks whence the gold of the different districts is derived, that its quality or fineness deteriorates the further north we go in New South Wales; in other words, it contains more silver and less gold.

"Thus the average fineness of Victorian gold is about 23 carats, that is to say, it contains about 96 per cent. of gold and 3½ of silver, with ½ per cent. of base metals; while, on passing north, we find the average fineness of New South Wales gold to be only 22 carats 1⅞ grains, or to contain 93½ per cent. of gold and 6 per cent. of silver. On going still further north to the colony of Queensland, the average fineness is little more than 21 carats (considerably below standard), or it contains 87¼ per cent. of gold and 12 per cent. of silver, and some contains only 85 per cent. of gold.

"These are averages only. It is not to be supposed that there is a regular and consecutive diminution in fineness with every degree of latitude we go north. There are exceptional localities in the north of this colony where gold is found of a high degree of purity, as at Rocky River, where it is over 23 carats fine, or 96 per cent. Possibly at a future time our geologists may be able to throw some light on these curious formed being more volatile than the chloride of copper, but much less so than those of lead, tin, zinc, or antimony.

"The method of refining now to be described is based upon these facts. It consists simply in passing a current of chlorine gas through the gold *while in a melted state*, which is easily done by thrusting into the molten metal a small clay tube connected with a stoneware vessel in which chlorine is generated. The chlorine on coming in contact with the silver in the molten alloy at once combines with it, forming chloride of silver, which, being of less specific gravity, rises to the surface of the melted gold, while the latter remains in a purified condition beneath. Chloride of silver has always been considered a somewhat volatile substance, and under circumstances such as those here described it was naturally supposed that it would be either sublimed in the flue or escape entirely up the chimney; but in practice it is found that the volatility of the chloride is not nearly so great as might have been anticipated, and that if it is coated with a layer of fused borax it may be kept melted at a high temperature without any very material loss. The furnace required for the operation is the ordinary 12-inch square gold-melting furnace, the principal points to attend to in its construction being—

"1. That the flue should be as near the top as possible, so as to allow of the crucible standing high up in it without being cooled by the draught.

"2. That the furnace itself should not be too deep, so that when the pot is placed in the fire the bottom of it may not be more than 3 in. above the bars. The covering of the furnace should consist of two fire-tiles, 7½ in. wide and 15 in. long, one of which should have a long slot or hole in its centre for the clay chlorine pipes to pass through. An iron cover will not answer, as it soon becomes much too hot for convenient working.

"The crucibles in which the refinage is performed should be French white fluxing pots* (made by De Ruelle, Paris); ordinary black-lead pots will not answer, owing to the reducing action they exert on the compounds formed. To prevent the infiltration of the very fluid chloride of silver into the pores of the clay pots, they are prepared by filling them with a boiling saturated solution of borax in water, which is allowed to stand in them for ten minutes and is then poured off, the crucibles being afterwards set aside to dry; the borax forms a glaze on the inner surface of the crucibles when they become hot in the furnace. When used for refining these French clay crucibles are placed within black-lead pots as a precaution against loss should the former crack, which, however, seldom happens. The crucibles are covered with loosely-fitting lids with the requisite holes bored through them for the passage of the clay chlorine pipes, &c. Ordinary clay tobacco-pipe stems, from 17 to 22 in. long, have been found to answer well for the purpose of passing the chlorine gas through the melted gold. Of late a pipe made in London to order, ½ in. in diameter, 22 in. long, and 5/16 in. bore, has been found to answer all requirements. The chlorine generators should consist of the best glazed stoneware acid jars, capable of holding from ten to fifteen gallons, and furnished with two necks. One of these openings should be stopped with a sound cork or vulcanized india-rubber plug if obtainable, through which should pass tightly two glass tubes—the eduction tube and the safety or pressure tube, the length of the latter 8 or 10 ft., and the former being a few inches spliced where necessary by means of vulcanized india-rubber tubing. The other opening, intended for introducing the oxide of manganese, &c., should be closed with a leaden plug, covered with a short piece of india-rubber tube by way of a washer, and well secured.

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* Special crucibles for the purpose are now manufactured at the Battersea Works, London, for the Sydney Mint.—M. E.

"Each generator should be charged with a draining layer of small quartz pebbles, down nearly to the bottom of which the pressure tube should extend. On this layer should be placed from 70 to 100 lbs. weight of binoxide of manganese in grains about ¼ in. cube, sifted from powder; this quantity will be sufficient to effect many refining operations, and will obviate the necessity of repeated dismantling of the apparatus. Each generator should be suspended to about half its height in a galvanized iron water-bath.

"The chlorine gas is produced when required by pouring common hydrochloric acid, specific gravity 1·15, down the safety tube, the apparatus being warmed by means of gas burners beneath the water-baths. The gas is conveyed from the generators by means of a leaden pipe fitted with branches to supply the several furnaces, all intermediate connections being formed by means of vulcanized india-rubber tubing, which, if screened from the direct radiation from the fire, stands the heat well, even immediately over the furnaces. All joints between the various pipes and india-rubber tubes are easily secured and rendered perfectly gas-tight with a cement consisting of a thin solution of india-rubber in chloroform.

"Screw compression clamps on the india-rubber tubes give the means of regulating the supply of gas as required, and enable the operator to shut off entirely so soon as the refining is over. The chlorine then having no means of escape accumulates in the generator, and soon forces all the acid up the safety tube into a vessel placed above to receive it, and the acid no longer acting on the oxide of manganese, the supply of gas of course ceases. Two such generators as are here described, and three ordinary gold-melting furnaces, have been found capable of refining daily about 2,000 ounces of gold, containing about 10 per cent. of silver, between 9 A.M and 2 P.M.

"The French crucibles, say size 17 or 18, duly prepared with borax, having been placed in the cold furnace and slowly and carefully heated to dull redness, the gold, from 600 to 700 ounces to each crucible, is introduced and the fire urged until the metal is melted, the necessary generation of chlorine having meantime commenced by the introduction of a little hydrochloric acid poured down the safety tube into the generators.

"As soon as the gold is melted from 2 to 3 ounces of borax in a state of fusion is poured upon its surface. If the borax is added sooner it acts too much on the lower part of the pot, and if thrown in cold is apt to chill the gold. The clay pipe which is to convey the chlorine to the bottom of the melted gold is now introduced. It is necessary to carefully heat the lower portion of this pipe for some ten minutes before introducing it into the molten gold, or it is apt to split. At the moment of its entering the melted gold the screw compression clamp is slightly loosened so as to allow a small quantity of gas to pass through it, and thus prevent any metal rising and settling in the pipe, which is then gradually lowered to the bottom of the molten gold, where it is kept by means of a few small weights attached to the top. The compression tap is now quite relaxed, and the gas is heard bubbling up through the melted metal, which it does quietly and without projection of globules from the pot.

"Sufficient hydrochloric acid must be added to the generators from time to time to keep up a rapid evolution of chlorine. A rough general rule is to allow one imperial quart of acid of 1·15 specific gravity to every 10 ounces of silver in the alloy operated on. The column of liquid in the safety tube, acting as it does like a barometer, affords a ready means of knowing the pressure in the generator, and of judging of the rate of production of the gas, as well as at once showing by its fall if anything irregular has occurred, such as a leak or a crack of the chlorine pipe or pot. From 16 to 18 in. in the safety tube correspond to and balance 1 in. of gold in the refining crucible. When the chlorine is first introduced into the melted gold, a quantity of fumes are seen to pass up from the holes in the crucible lid; these are not chloride of silver, but the volatile chlorides of some of the baser metals, and they are especially dense when much lead is present in the alloy under treatment, forming a white deposit on any cold substance presented to them. After a time, longer or shorter according to the impurities in the gold, the fumes cease. So long as any decided quantity of silver is present in the molten gold, the whole, or nearly the whole, of the chlorine is absorbed, little, if any, appearing to escape and to be thus wasted, and it is found that the better the supply of chlorine the quicker is the operation.

"It is a curious circumstance that though, in toughening with corrosive sublimate, this substance is only thrown on the surface of the melted gold, yet the whole mass is toughened by its action. It seems essential, in using chlorine, that the gas should pass to the very bottom to effect a complete refinage.

"As soon as the operation is nearly over fumes of a darker colour than those observed at the commencement make their appearance, and the end of the refinage is indicated by a peculiar flame or luminous vapour of a brownish yellow colour, occasioned by the free and now waste chlorine escaping, which can be seen on removing a small plug which fits into a hole in the lid of the pot. This, however, of itself is not a sufficient indication; the process is not complete until this flame imparts to a piece of white tobacco pipe, or similar substance, when held in it for a moment, a peculiar reddish or brownish yellow stain; so long as it gives any other colour the refinage is unfinished.

"When these appearances are observed, usually for gold containing about ten per cent. of silver in about an hour and a half from the introduction of chlorine, the gas is shut off and the pots removed from the fire, the white crucible lifted out of the black one, and together with its contents allowed to stand several minutes, until the gold becomes cool enough to set or solidify. The chloride of silver which remains liquid much longer is poured off into iron moulds. The crucible is then inverted on an iron table, when the still red-hot gold falls out in the shape of a cone; this is slightly scraped and then thrown hissing into a concentrated solution of common salt, to free it from any adherent chloride of silver.

"An alloy containing originally 89 per cent. of gold, 10 per cent. of silver, and 1 per cent. of base metals, will yield on an average a cake of chloride weighing, with a little adherent borax, 15 ounces for every 100 ounces operated on.

"It is necessary very carefully to dry and heat the moulds into which the chloride of silver is poured, as the slightest moisture causes the latter to be violently dispersed while red hot, to the great risk of the bystanders.

"With ordinary care this will never happen, but attention is called to the point as a very deliquescent chloride of iron is apt to form on the moulds.

"The gold is now fine, and simply requires re-melting into ingots.

"As before stated, it is found that all these operations can readily be performed, and about 2,000 ounces refined per day in three common melting furnaces, in five hours; 98 per cent. of the gold originally contained in the alloy operated on is then ready for delivery. The other 2 per cent. remains with the chloride of silver, partially in the metallic state and partly in a state of combination with chlorine, and probably with silver.

"To free the chloride of silver from this combined gold it is melted in a boraxed white-pot, with the addition of from 8 to 10 per cent. of metallic silver, rolled to about one-eighth of an inch thickness. The chloride of gold is by this means reduced at the expense of the metallic silver, chloride of silver being formed, while the liberated gold sinks, and melts into a button at the bottom of the pot. As soon as the whole is thoroughly melted the pot is removed from the furnace and allowed to stand about ten minutes, and the still liquid chloride of silver is then poured into large iron moulds, so as to form slabs of convenient thickness for the next operation, that is, the reduction to the metallic state.

"After the fusion of the chlorides a small quantity of a curious spongiform substance adheres to the sides of the crucible used, probably consisting of sub-chloride of silver, but since it always contains a little gold care has to be taken in pouring off the fluid chlorides to prevent this auriferous sponge from falling out and mixing with them.

"The slabs of chloride of silver are reduced without difficulty by plates of wrought iron or zinc in the usual way. Besides the separation and recovery of the silver as above described, another useful end is gained by this process.

"A very large proportion of the gold of Australia is more or less brittle, an effect generally due to the presence of small quantities of lead or antimony, rendering the bullion quite unfit for coinage or manufacture until it has undergone some process to render it tough.

"The methods usually employed for this purpose are either fusion with nitre and borax, melting with oxide of copper, or the addition of corrosive sublimate (bichloride of mercury) to the melted gold. The two former of these plans are troublesome, from the corrosive action they exert on the crucibles, and the last, namely, the employment of corrosive sublimate, which is that usually employed, is most objectionable, from the dense and highly injurious fumes evolved.

"The passage of chlorine gas through the melted gold is found to effect the complete toughening of the metal by the elimination, as volatile chlorides, of the materials which render it brittle, while the evolution of the deleterious mercurial fumes is avoided.

"In the metallurgic treatment of the precious metals some loss is always sustained, but that incurred in the process here described is found very small.

"The average loss of gold in operating hitherto has been found to amount to 19 parts in every 100,000 of alloy treated, which is considerably less than would be met with in toughening an equal amount of gold with corrosive sublimate in the ordinary manner.

"The loss of silver has amounted to 240 parts in every 100,000 of alloy operated on (containing originally, say, 10 per cent of silver). There is no doubt that a considerable portion of both these losses would be recovered on further treating the pots and ashes remaining after the operation; and it is found that, as manipulatory skill is acquired, the proportional loss of silver appears to be decreasing. In refining on the large scale, gold containing 10 per cent. of silver, the cost of the operation in Sydney is about five farthings per ounce. The fineness of the gold produced by this process varies from 991 to 997 in 1,000 parts, the average being 993·5 or 23 carats 3⅔ grains. The remaining 6½ thousandths are silver, and this compares favourably with any of the previously known practical processes, none of which leave less silver than this in the resulting fine gold.

"If the refined gold be subjected to a re-refinage by chlorine, the amount of silver left in it can be reduced to 0·2 per cent., just as in the refinage by the ordinary sulphuric acid process, the same result can be obtained by subjecting the refined gold to a further refinage by bisulphate of potash. For practical working, however, this would probably never be attempted.

"The silver resulting from this method of refining is tough, but its quality varies somewhat according to the gold originally operated on; if the alloy treated contains much copper, the greater part of this remains with the resulting silver, but the other metals are nearly all eliminated.

"The fineness of the silver hitherto obtained has varied from 918·2 to 992 in 1,000 parts, the average being 965·6. Analysis of the silver resulting from the refinage of gold known originally to have contained, amongst the base metals in the alloy, copper, lead, antimony, arsenic, and iron, gave the following results:—

Silver	972·3
Copper	25·0
Gold	2·7
Zinc and Iron	Traces.
	1000·0."

CHAPTER IX.

THE MELTING AND ASSAYING OF GOLD.

MELTING AND ASSAYING—Fusion with Borax—Melting Furnaces—The Molten Processes of an Assay—Manipulation of Weighing the Assay Piece—Valuation of Gold Alloys—Preparation of the Assay Piece—Cupellation—Parting of the Assay—Professor Roberts-Austen on the Parting Process—Parting Assays—Gold containing Oxidisable Metals—Assaying Gold Quartz—Assaying by Scorification—Forms of Cupels—Weighing the Button of Precious Metal.

Fusion with Borax.—When the gold is received from the gold washer, in the shape of gold dust, or when it is separated from its ores by amalgamation or otherwise, it has to be subjected to a series of metallurgical operations to obtain it in a state of perfect purity. The first of these is that of fusion with borax, and casting the metal into ingots, by which it is obtained in a convenient form, and freed at the same time from earthy and other impurities. For this purpose the gold dust from the washings, or the spongy gold obtained by amalgamation, is mixed with a little dried borax, and is introduced into a black-lead crucible, Fig. 110, previously heated in a common melting furnace.

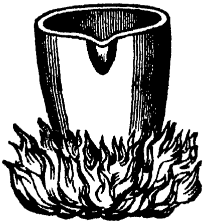

FIG. 110.—PLUMBAGO CRUCIBLE.

Melting Furnaces.—A convenient furnace is shown in Fig. 111. It consists of two cylinders of sheet iron, placed one within the other. A represents the outer and B the inner cylinder—the latter lined with a coating of fire-clay about one inch thick. Both cylinders are provided with a bottom, and are fixed together at the top, air-tight, by a horizontal hoop or flat ring of metal, E E, in such a way as to leave an equal space, C C, between their sides and bottom. The interior cylinder, B B, is pierced at about the middle of its depth with eight holes, D D, which pass through the lining of fire-clay, and all point to the centre of the furnace, where the crucible, H, is placed on a piece of fire-brick, kept in its position by a little fire-clay, and surrounded with fuel. For a small furnace of this kind charcoal must be employed as fuel, and should be broken into pieces about as large as a walnut. The air is blown into the opening, F, by means of a double-action bellows, or a simple rotating fan may be used. By this means the air is driven in a steady equable current through the holes, D D, into the cavity of the furnace. The heat which may thus be produced with a furnace having an internal diameter of only a few inches is so intense as to be capable of melting manganese, or several ounces of cast-iron, with the greatest ease.

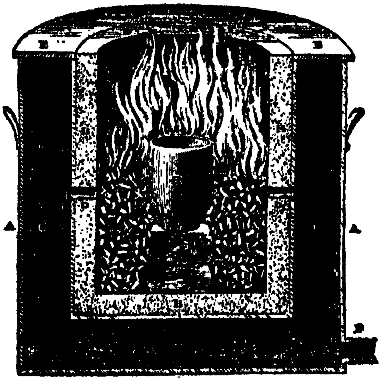

Fig. 111.—Melting Furnace.

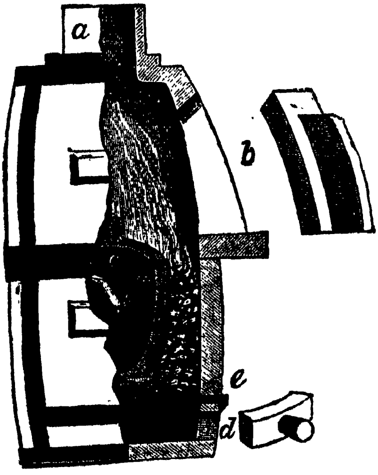

Fig. 112.—Portable Wind Furnace.

A very convenient furnace for the fusion of small quantities of gold bullion up to 100 or 150 ounces, and for the melting of ore assays, is shown in Fig. 112. Being made in sections, it is easily packed, and is of simple and durable construction, being built up of fire-clay and strengthened with hoop-iron bands. In the one which I use in my laboratory I can place with ease four Battersea No. 10 clay crucibles for ore assays. Owing to its portability this furnace is specially adapted for use in distant mining regions. It is made in three sizes, namely: B, 10 inches diameter, 20 inches height, 67 lbs. weight; C, 11¼ inches diameter, 22 inches height, 90 lbs. weight; and D, 13½ inches diameter, 26 inches height; 144 lbs. weight. In the illustration, *a* is the socket to fix on iron chimney; *b* the door for putting in crucible, and for cooking; *c* the crucible; *d* the door for regulating the draft; *e* the iron grate.

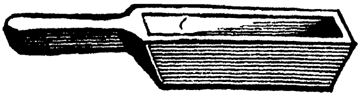

Fig. 113.—Ingot Mould.

After the fusion of the metal, a slag, consisting of the grosser impurities, accumulates on the surface; this, when too fluid, must be thickened by the addition of a little bone-ash, and then skimmed off. The metal is now cast into ingots by being poured into iron ingot moulds (shown in Fig. 113), previously warmed and oiled by wiping the internal surface with a piece of tow dipped in sweet oil.

Fig. 114.—Crucible Tongs.

The removal of the melting pot from the furnace with the crucible tongs (shown in Fig. 114), as well as the pouring the molten gold into the mould, requires a certain amount of practice and dexterity, as the operation must not allow time enough to chill the gold in the pot when once it is out of the fire. When the quantity of gold is large, the molten mass ought to be stirred in the pot when fused, and just before pouring. An iron rod is heated at one end to bright redness and then introduced into the molten mass and stirred gently, which insures a perfect uniformity of the ingot when cast.

The black-lead crucibles which have been used in a series of melting operations retain some globules of gold, and these are preserved with a view to the subsequent extraction of the precious metal from them.

When gold contains certain impurities, if they consist of easily oxidizable metals, these are generally removed by the addition of saltpetre in the last stages of the melting operation, which generally exerts a powerful oxidizing action on some of the so-called base metals.

Several Processes of an Assay.—The assaying of gold means its quantitative determination in an alloy. The following metallurgical method is adopted in the Royal Mint and other establishments where assaying is regularly practised, it being not only more expeditious than the wet analysis, but admitting also of a greater number of assays being simultaneously conducted. The method consists of the following processes:—

(1.) The accurate weighing and preparation of the assay piece or sample. In case of gold ingots two of the corners are chipped off with a chisel, one from the top and one from the bottom.

(2.) Cupellation, or removing the copper and other base metals by oxidation or absorption.

(3.) Parting by inquartation, or separation of the silver from the gold by dissolving the former metal in nitric acid.

(4.) Final weighing of the resulting "cornet," and the applying of necessary corrections.

Weighing the Assay Piece.—It is evident that, as the amount of metal to be operated upon is extremely small, it is of the utmost importance that the weight of the assay piece should be very accurately determined, both before and after the process. For this purpose very delicate balances are used. The form most generally approved* is shown in Fig. 115, which is capable of indicating $\frac{1}{20000}$ part of an assay pound of half a gramme (7·716 grains). The balance employed must be extremely sensitive, sure, and quick—qualities which require a very light and carefully constructed beam. The beam in the balance shown is 10 in. long, and usually weighs about 125 grains. The scale pans rest in stirrups of palladium, and can be removed by means of forceps. The supports at the centre and ends are shown on a large scale in Fig. 116. Each stirrup hangs from two steel points resting in agate cups, and the agate knife edge at the centre does not come in contact with the agate palates until the supports have been removed from below the scale pans by means of the handle moving in a vertical slot in front of the balances.

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* As made by Becker, of New York, and by Oertling, of London, and other firms.

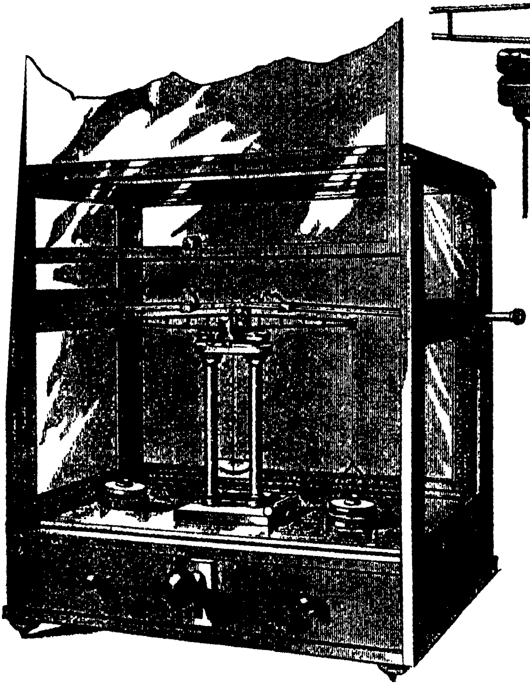

FIG. 115.—ASSAY SCALES.

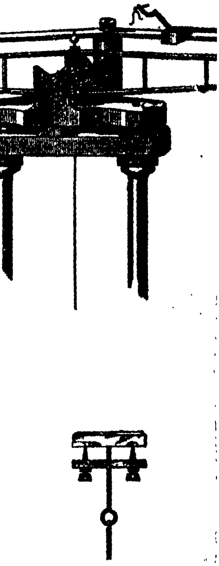

FIG. 116.

Two systems are employed for expressing the composition of gold and silver ingots, and the weights used in "weighing in" and "weighing out" the assays are usually so prepared that the results obtained directly represent the composition, rendering calculations unnecessary. Thus when reports of gold assays are required on the trade system, the weight of metal employed (varying from 5 to 16 grains)—called the "assay pound"—is divided into twenty-four parts, called carats, and each of these is subdivided in carat grains and eighths of carat grains.

If, however, assays are required on the millesimal system, the weight taken is called 1,000, and in weighing the cornet after the completion of the assay subdivisions of this 1,000 are employed.

The 1,000 weight, taken as the assay pound, is generally ½ gramme, and is the centre piece of an ivory box, containing all the subdivisions on this weight on the millesimal system.

Valuation of Gold Alloys.—In England the standard of the alloys of gold is calculated in fractions of unity expressed in carats. Perfectly pure gold is taken as unity, and this is assumed to consist of 24 fractional parts or carats, each of which is divided into 4 imaginary grains, and these are again subdivided into eighths and "excess grains," so called in contradistinction to the primary divisions of a carat or "carat grains." One carat grain contains 60 excess grains; thus, the divisions of the original weight or "assay pound" taken will be as represented in the following table:—

Assay Pound.	Carat.	Carat Grain.	Eighth.	Excess Grains.	Decimal Equivalent. Per Cent.
				1	·017
			1	7·5	·130
		1	8	60	1·042
	1	4	32	240	4·167
1	24	96	768	5760	100·000

From the above the meaning of the term "excess grain" will at once be evident. It is the $\frac{1}{5760}$th part of the weight of metal taken for assay, this ratio being the same as that borne by a troy grain to a troy pound. For accurately reporting the amount contained in a given alloy, it is necessary to use subdivisions less than eighths. "Excess grains" are therefore employed, and their number is the same as the number of actual grains of gold contained in a troy pound in excess of the amount represented by the report in carats, grains, and eighths.

The standard alloy of the English gold coinage contains 11 parts of gold to 1 of copper, or in other words, 22 parts of gold to 2 of copper; it is therefore said to be 22 carats fine. When assays are reported on this system, which is called the "trade system," the actual amount of gold is not given, but the metal is described as so much better or worse than standard, to indicate which the letter B or W is prefixed. Thus, suppose an assay to be given as B 1 ct. 3 grs. 5 eighths + 3·5 excess grains. This means that it actually contains 23 cts., 3 grs., 5 eighths, and 3·5 excess grains, in the total amount of 24 carats. The percentage composition of such an alloy would be 99·66 per cent. of pure gold and 0·33 per cent. of alloying metal (generally copper or silver).

This system is still retained in the Mint, but has beer replaced by a decimal system. Perfectly pure gold is termec "1000 fine," and the report upon any sample of alloy indicate the number of parts of pure gold or silver in 1000 which th sample contains. The English coin standard of 22 would b equal to 916·667. In France and in the United States, th standard coin is expressed by 900 of gold and 100 copper.

The following table, which gives the values of the division used in the old method expressed in decimal fractions, will b found convenient for effecting any required conversion in th case of gold. As an example of its use, suppose it be require to convert the above trade report into its equivalent on th decimal system. Then—

	Thousands.
23 carats will be equivalent to	958·22
3 carat grains ” ”	31·25
5 eighths ” ”	6·51
3·5 excess grains ” ”	0·61

Carat.	Decimal Equivalent.	Carat Grains.	Decimal Equivalent.
1	41·667	1	10·417
2	83·333	2	20·833
3	125·000	3	31·250
4	166·667	4	41·667
5	208·333		
6	250·000	Eighths.	Decimal Equivalents.
7	291·667		
8	333·333	1	1·302
9	375·000	2	2·604
10	416·667	3	3·906
11	458·222	4	5·208
12	500·000	5	6·510
13	541·667	6	7·812
14	583·333	7	9·115
15	625·000	8	10·417
16	666·667		
17	708·333	Excess Grains.	Decimal Equivalents.
18	750·000	1	0·174
19	791·667	2	0·347
20	833·333	3	0·521
21	875·000	4	0·694
22	916·667	5	0·868
23	958·222	6	1·072
24	1,000·000	7	1·215
		7·5	1·302

Preparation of the Assay Piece.—The corners of the ingots, if of malleable gold, are hammered out on a small anvil with a clean flat hammer, and rolled out into two flat strips between a pair of rollers (Fig. 117). This facilitates the assayer in cutting up his assay pound and weighing it out to a nicety. The shears used are shown in Fig. 118. If the gold does not contain about 2½ times its weight of silver, it is necessary to add silver up to this amount. For instance, if a bar is supposed to contain 700 parts of gold and 300 parts of silver, it is necessary to weigh out 1,450 parts of chemically pure silver, and add this to the assay pound. If no copper is present in the alloy, a very small piece of copper weighing about 30 milligrammes is added, to prevent brittleness in the alloy in the subsequent lamination; but if copper is in the alloy no addition is required. The two metals are then wrapped up in a piece of finely rolled sheet lead, and placed on a tray divided into numbered compartments, which correspond to the positions of cupels in the muffle; the number of the compartment in which each assay is placed being noted with a view to its subsequent identification.

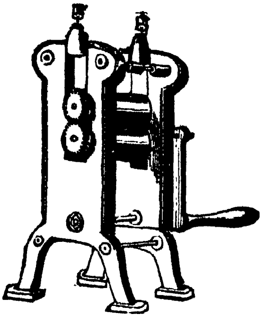

FIG. 117.—ROLLERS.

Cupellation.—The assays having been thus prepared are now ready for *cupellation*. A certain number of cupels are placed in a muffle and heated there to redness. The assays having been placed in the cupels, the opening of the muffle should be loosely closed by a piece of hot fire-brick, or by pieces of charcoal. After a short interval all the assay pieces will be melted, and in each cupel will be seen a round mass of molten metal, on the surface of which appear bright patches of litharge; these are produced by the oxide of lead which is constantly forming, and passing off from the centre to the edges of the assay, where it meets with and sinks into the porous matter of the cupel, leaving always a new surface exposed to the current of air which is carried through the muffle. At the same time there rises and passes through the openings of the muffle a white vapour which is caused by the volatilization of a portion of lead. When this vapour is very thin, and rises with great rapidity, it is an indication that the heat is too great, and part of the draught must be stopped; on the

FIG. 118.—SCISSORS OR SHEARS.

other hand, when it is thick and heavy, and hangs sluggishly over and around the cupel, the temperature is too low, and the draught must be increased.

As soon as the greater part of the lead has been thus converted into litharge and absorbed, the remaining bead of rich alloy suddenly becomes opaque and glowing, and is traversed by iridescent bands produced by the extremely thin films of fluid litharge resulting from the last traces of lead. This is a sign that the process is on the point of completion, and means must now be taken to increase the heat of the furnace for a short time, so as to insure the expulsion of the last remaining portions of lead. But if silver be present in the assay, and if it be desired to estimate also the amount of that metal, the increase of temperature towards the end of the process must be of short duration, otherwise a considerable portion of the silver would soon volatilize. Immediately after the disappearance of these bands the globule becomes steady, after which the peculiar action known as "brightening" takes place, when the metal is observed to emit a brilliant flash of light, and it immediately solidifies. If the cupellation has been well performed, the gold—or the mixture of the two precious metals—is now in a state of almost chemical purity.

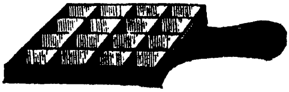

FIG. 119.—CUPEL TRAY.

When the cupellation is terminated, the mouth of the muffle ought to be closed, and the whole allowed to cool until the buttons become solid; or when only a few assays are performed at a time, the cupels containing the buttons of pure metal may be drawn to the front of the muffle, and allowed to cool slowly. They are then withdrawn, and placed in numerical order on the cupel tray (Fig. 119), which avoids errors being made. Some assayers mark and count the cupels from the right, or the reverse, and after some practice mistakes never happen. This refrigeration must not be too rapid, otherwise there may be sensible loss by "spitting, sprouting, or vegetation," an effect, however, only produced when a large proportion of silver is present. To insure gradual cooling, it is occasionally convenient to invert over the cupel containing the bead another heated cupel. Some assayers remove the cupels from the furnace while the metal is in a molten state, but the practice cannot be recommended, for besides the danger of upsetting a cupel and the "spitting" of the assay the buttons are found more malleable when cooled gradually.

The button is detached from the cupel after cooling by a pair of steel forceps, cleaned with a brush formed of stiff bristles, and flatted as presently described. If the process has been successful the button should be round and bright upon its upper surface, but somewhat rough or crystalline on the part by which it was attached to the cupel, from which it ought to be removed without difficulty.

The presence of iron, tin, nickel, or zinc, in the alloy under assay materially interferes with the success of the operation, as these metals will not pass into the cupel by the aid even of a large quantity of lead, but from their rapid oxidation will either volatilize, or their oxides will accumulate on the surface as a sort of slag, in which particles of precious metals may be entangled. In this case the assay becomes what is technically termed *foul*, the cupel not being able to absorb the metallic oxides as they are formed. To obviate this evil, the preliminary process of scorification should be resorted to, when the presence in the alloy of any of the above metals is known or suspected. A convenient weight of native gold or rich alloy to operate upon is 10 to 15 grains.

The lead employed must be in all cases free from silver, being such as has been derived from pure litharge, otherwise it would be quite impossible to estimate the true composition of the alloy.

The assay of gold furnishes results which are more accurate than those obtained in the cupellation of silver, the loss of gold by volatilization being very much smaller, and scarcely any of the metal is carried into the cupel by the lead.

Parting of the Assay.—After the preceding operations, the button of alloy which remains is composed of gold and silver—not indeed absolutely pure, for whatever precautions may be used it will still retain a small quantity of lead, and frequently also traces of copper. For practical purposes, however, it may be regarded as a pure alloy of the two precious metals, and in this view it now only remains to separate the gold from the silver.

It has been stated that when silver and gold are intimately mixed together, as in an alloy, it is necessary that the silver should be present in the proportion of at least 2½ parts to 1 part of gold, otherwise its particles will be so enveloped and protected by those of gold that the nitric acid will be prevented from exercising its solvent action upon them. Care should always be taken before the introduction of the assay piece into the furnace that the requisite proportion of silver is present, and if this be the case it will be sufficient at once to subject it to the action of the acid.

But if the silver be present in a less or much greater proportion, then, as a preliminary step to the operation of parting, the alloy must be brought to the proper standard, either by the addition of the necessary quantity of pure silver, or by adding more gold, as the case may be ; for if the proportion of silver exceed 3 parts, the gold will be precipitated by the acid as a dark powder, instead of retaining its compact form, and thus the accuracy of the estimation of the amount of gold present will be impaired.

It is hardly necessary to add that the amount of silver requisite can only be determined when the approximate assay is known. This may be determined by a wet analysis, a rough assay, or by the touchstone ; or experienced assayers are able to determine with sufficient accuracy by observing the colour and the facility with which the metal can be cut with shears.

The button is now flattened by hammering on an anvil, three strokes being so given that the first shall extend it to about the size of a sixpence, and the other two reduce the thickness still further at two opposite edges, in order to facilitate its introduction into the laminating mill. These flattened buttons are generally annealed before rolling, and by the rolling operation the thickness is reduced to about one-eighth of an inch, and when all the laminæ have been placed in slits on a wrought-iron tray, they are introduced into the muffle and annealed. The operation renders them sufficiently flexible to allow of their being twisted into a small spiral cornet, by rolling between the finger and thumb, or by means of a pair of round-nosed pliers.

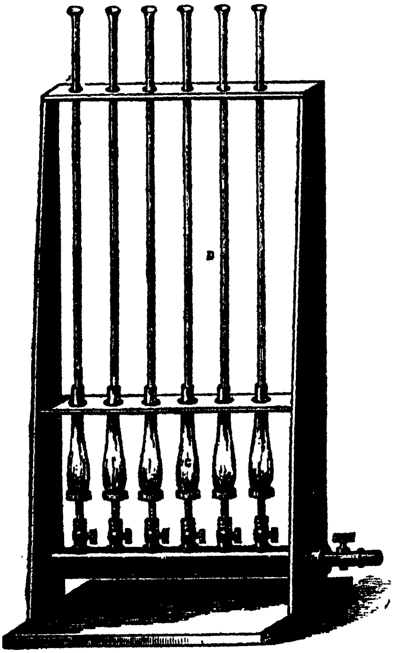

FIG. 120.—PARTING FLASK.

To effect the parting of the assay, each assay piece, or cornet, is boiled separately in strong nitric acid, which must be free from chlorine, sulphuric acid, or sulphurous acid. The nitric acid is contained in a small glass flask, called a parting flask. Fig. 120 represents a convenient apparatus for boiling six cornets separately. The large tube, A, is connected with the gas supply, and from it rise a number of small rose Bunsen burners. On these rest the parting flasks, C, into each of which is introduced from 2 to 3 oz. of nitric acid of 1·26 specific gravity. The long tubes, D, dipping into the mouths of the flasks, are intended to condense as far as possible the acid vapours, the condensed liquid running back into the flasks. In a case where a large number are in action at once, they should be connected with a flue to carry off these vapours. The acid should continue to boil from three to five minutes after the glasses have cleared of fumes; the acid is then poured off, the cornets washed with hot distilled water, and again boiled from fifteen to twenty minutes in nitric acid of about 1·30 specific gravity. As the ebullition is at times very violent, and the cornet, owing to its spongy nature, extremely fragile, it is generally recommended that a piece of charcoal be introduced into each flask. This, however, is objectionable, as it may cause the formation of nitrous acid, which by absorption in the acid will even dissolve portions of the metal. This source of error is completely eliminated by using instead of the charcoal small balls of porous earthenware, as recommended by Field.

The last acid having been poured off, the flask is completely filled with distilled water, and a small crucible of smoothly-finished porous clay is placed over the mouth of it. The two vessels thus adjusted are then inverted, so as to allow the cornet to fall gently through the water into the crucible, and by a dexterous movement of the hand the flask is withdrawn, in such a manner as to prevent the overflow of any liquid from the little crucible. The water is now carefully decanted, and as much moisture removed as possible; the crucible containing the cornet is then heated to redness in the muffle. Under this final heating it is not fused, but shrinks in bulk, loses its brown appearance, and assumes the peculiar colour and lustre of gold; while at the same time it is rendered more compact, less brittle and fragile, and when cold can be removed by a pair of forceps to the scales, in which it is weighed with the same precision and care as the original alloy.*

Professor Roberts-Austen, F.R.S., in a lecture delivered before the Society of Arts on March 31st, 1884, gave the following interesting data respecting the parting process:—

"This action of nitric acid on an alloy of gold and silver was certainly known to Geber and the early alchemists, but the first official mention of the use of the parting assay appears to be in a decree of Philippe de Valois, in the year 1343, confirming its use in the French mint. There is, then, this difference between the assay of gold and silver. In the case of the cupellation assay of silver, the button of metal has only to be removed from the cupel, and when the adhering bone-ash has been removed by the brush it passes direct to the balance.

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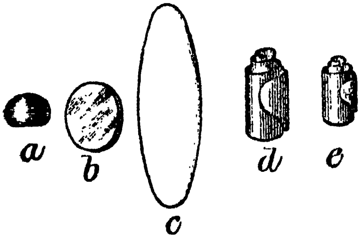
* Materials required for assaying may be obtained of Messrs. John Orme and Co., 65, Barbican, London, E.C.

"After the first stage of the assay of gold on the cupel, the object to be attained is mainly to secure a button, *a*, of gold and silver in a convenient form for submitting to the subsequent operations. The alloying stage would, however, be just as effective if it were conducted in a small non-porous receptacle, such as a small crucible of glazed porcelain.

"The subsequent operations are, flattening the button, *b*, annealing it, rolling it into a strip, *c*, and annealing it a second time. It is then coiled into a spiral, or cornet, *d*, and treated by two successive portions of nitric acid, in order to remove the silver; after this the spiral of spongy gold, *e*, which retains the original form given to the silver alloy, is heated to redness, when it becomes bright, and is sometimes so coherent that it may be unrolled without fracture."



Scale about ½.

FIG. 121.—BUTTON AND CORNETS.

The weight of the cornet will indicate very nearly the amount of gold in the alloy if the assay has been properly conducted, but not with absolute exactness. Even when every precaution has been used, the gold contained in the cornet is never chemically pure; it still retains a very small quantity of lead and silver, and frequently also traces of copper, in consequence of which the weight of the cornet may exceed by $\frac{1}{2000}$th part the true result; and when it is considered that in assaying alloys the quantity taken is often not more than 7 or 8 grains, it will be seen that even this small error might lead to serious disappointment in estimating the proportion of gold in large masses of bullion. To ascertain the amount of the error, and thus arrive at a perfectly correct estimate, it is usual to assay, simultaneously with the alloys, a number (generally three) of what are termed *proofs*, consisting of weighed portions of perfectly pure gold, approximately equal in amount to that present in the assay pieces. The cornets obtained from these proof assays are weighed, and the excess or deficiency in weight over that of the gold which was introduced indicates the amount of correction which it becomes necessary to apply to the other assays. This correction is liable to vary, according to the temperature of the furnace and other causes; but it usually ranges from 0·2 to 0·5 in 1,000. It may be observed that it is not essential to employ *absolutely* pure gold, provided the difference from purity be known. Thus the corrections to be applied to a gold assay will usually be one-half of a thousand.

The actual amount of metal retained—called surcharge—is greatly influenced by the amount of copper present in the alloy under examination, and varies considerably with the heat of the furnace and other circumstances. In an alloy rich in gold, which requires therefore only a small quantity of lead, the surcharge is slight; when, however, a large quantity of copper is present the surcharge becomes negative, in consequence of the absorption of gold by the cupel; and for alloys containing 68 or 70 per cent. of gold the surcharge is zero. Rossler has shown that the gold lost in the cupellation process increases with the amount of lead used, and decreases as the silver present is increased; and these conclusions are confirmed by the results of experiments recently made in the Royal Mint. He also considers that the gold lost is generally greater in amount than the silver retained by the cornet.

Professor Roberts-Austen gives the following formula for the corrections to be applied to a gold assay when checked by means of a *proof* or *check*:—Let 1,000 be the weight of alloy originally taken; p, the weight of the piece of gold finally obtained; x, the actual amount of gold in the alloy expressed in thousandths; a, the weight of gold supposed to be absolutely pure taken as a check, which approximately equals x; b, loss or gain in weight experienced by a, during the process of assay; k variation of "check gold" from absolute purity, expressed in thousandths. Then the actual amount of fine gold in the check-piece $= \text{A}\,(1-\frac{\text{K}}{1{,}000})$, and the corrected weight will be $X = P - \frac{AK}{1,000} \pm 6$, the 6 being added or subtracted according as it is a loss or gain.

If (A) be assumed to be equal to (X) this equation becomes

$$1 + \frac{K}{1,000}$$

Parting Assays.—It is frequently necessary to determine not only the amount of gold or silver contained in a given alloy, but also the amount of each of these precious metals present in the same sample. The assays made with a view to obtain this information are termed "parting assays."

Gold and Silver.—If the quantity of silver in the alloy submitted to the operation of parting be more than three times that of gold, the separation may still be effected by nitric acid, but the gold will be left in a state of powder. It may, however, be washed, ignited, and weighed. Should the amount of silver be small compared with the gold, and the assayer wishes to avoid the trouble of inquartation, that is, the adding of the requisite proportion of silver, the alloy may at once be subjected to the action of aqua regia, when the gold will be dissolved, and the silver thrown down as an insoluble chloride. The clear solution containing the gold is then decanted, and when the excess of acid has been expelled by evaporation, the gold can be precipitated by sulphate of iron.

The determination of the amount of silver contained in gold by the dry method of assay is a somewhat complex operation. One or two gold assays in the ordinary way are made, and at the same time two more assay pieces are passed through the furnace, but without the addition of silver. By this means all the oxidizable metals are in both cases removed, and the difference in weight between one of the cornets resulting from the ordinary assay, and the button of mixed gold and silver as drawn direct from the furnace, must be due to silver originally present in the alloy.

The silver contained in the solution of its nitrate resulting from the common process of parting, may readily be recovered by adding an excess of chloride of sodium, which will throw down the whole of the silver as white insoluble chloride. This precipitate, which speedily becomes slate coloured on exposure to light, is collected and put into a vessel containing water acidulated with hydrochloric acid, some zinc or iron is introduced, and the silver is rapidly restored to its metallic state, in the form of a dark grey powder.

The supernatant liquid having been decanted, this fine metal is washed with dilute hydrochloric or sulphuric acid, in order to remove traces of iron or zinc; but as the removal of the former of these metals may be effected more perfectly, it is now always used in preference to the latter; after washing with water, and drying, the silver is fused in a crucible with the addition of a little carbonate of soda or nitre, and cast into bars.

Gold and Copper.—The alloys of gold and copper are cupelled like the alloys of gold and silver; but as copper has a very great affinity for gold it is necessary to use a larger proportion of lead to insure its oxidation when combined with gold than when united with silver. This proportion varies according to the standard and the temperature. It is admitted that for the same standard there must, under similar circumstances, be twice as much lead used in the cupellation of gold as in that of silver. Thus 14 parts, at least, ought to be employed in common furnaces for an assay of gold coin which contains 0·100 of copper. There is no inconvenience in employing a little more, as it does not increase the loss of gold. However great the proportion of lead may be that is added to the cupreous gold for the purpose of cupellation, the button always retains a very small quantity of copper, of which a fresh cupellation does not free the gold, and which occasions what is termed the surcharge. This surcharge being very slight it can be neglected in assays of minerals, but it is necessary to take notice of it in the assays of alloys. But it is known that the presence of silver much facilitates the separation of copper from gold, and it is rare that an alloy of cupreous gold does not contain a little silver, which must be separated; and when this is not the case a small quantity of that metal can be introduced into the alloy, so as to be in about the proportion of 2½ parts to 1 of gold. When an assay is to be made of an alloy of gold and copper, a sufficient quantity of silver should be added to fulfil the condition according to the presumed standard, which is approximately determined by a preliminary assay and then cupelled with lead.

TABLE FOR PROPORTION OF LEAD TO BE EMPLOYED IN THE CUPELLATION OF GOLD AND COPPER.

Gold in Alloy.	Lead required.
1000 thousandths	1 part.
900 ,,	10 parts.
800 ,,	16
700 ,,	22
600 ,,	24
500 ,,	26
400, 300, 200, 100, 50 thousandths	34

Gold containing Oxidizable Metals.—Where gold contains platinum with copper, or other oxidizable metals, the latter will be removed with difficulty by cupellation, and silver must be added. Double the weight of the assay pound in silver must be taken, and the cupellation must be conducted at a high heat on the addition of the proper quantity of lead.

Mr. Chaudet, who has made experimental researches on the alloys of gold, silver, copper, and platinum, states that the copper can be more completely separated, and less silver lost, by cupelling at a high temperature, with the least possible quantity of lead, than by employing more lead and working at a lower temperature. He states that it is almost impossible to separate all the copper, and no advantage can be obtained by increasing the quantity of lead. When almost the last traces of the copper are separated the button must be cupelled afresh with a small quantity of lead; but a small quantity of silver is nearly always lost. In all cases, in order that no lead shall remain, it is necessary to leave the assay button some few minutes in the muffle, after the cupellation is finished.

The alloys of gold and silver containing platinum exhibit certain characteristics during cupellation, or parting, which indicate the presence of this metal. If there is only two per cent. of platinum in the alloy, the button remains flat on the cupel, and if this button is dissolved in nitric acid, the solution is coloured straw yellow. The peculiar iridescence of the button at the termination of the assay is not as bright and lively, and is much longer, than when no platinum is present; and on solidifying the button remains dull and tarnished, and does not exhibit a bright lustre with a partly crystalline structure. When the alloy contains more than a hundred parts of platinum to nine hundred of gold, the cornet after annealing is of a pale yellow colour, tarnish silver grey.

The following method serves for detecting platinum in admixture with gold, and other heavy matters obtained by washing auriferous gravels, or in prospecting: If gold is present, collect it with mercury; take the residue and boil it in aqua regia, evaporate to dryness, add some dilute hydrochloric acid, boil and filter; to the filtered solution add a strong solution of sal-ammoniac. If a bright yellow, or reddish yellow, granular precipitate falls, platinum is present in the sand.

Auriferous bismuth and lead are either directly cupelled, or, with a very small percentage of gold, first scorified and several buttons concentrated to one.

Auriferous tin is oxidized in the muffle, and scorified with the addition of granulated lead and borax, and then cupelled.

Auriferous iron, steel, or pig iron is oxidized with nitric acid, the resulting fluid evaporated to dryness, and the dry mass well scorified with eight to twelve parts of lead, two to three parts of borax, and one part of glass.

Auriferous mercury is scorified sometimes after previous distillation with a very gradually increasing temperature, and with eight parts of lead.

Gold containing rhodium is fused with three to four parts of silver parted as usual with nitric acid, and the well-washed and dried gold fused at a red heat in a platinum capsule with bisulphate of potassa or soda, whereby the rhodium, together with the remnant of silver, is dissolved, with brisk evolution of sulphurous acid, and the formation of a brownish red to black salt. The fluid mass of salts is poured off from the gold, and the fusion with bisulphate of soda again repeated, whereupon the fluid salt appears but slightly coloured. After this is again poured off, the gold is boiled a few times with distilled water, dried, and ignited. In order to be certain of the complete removal of the rhodium, the gold is again mixed with three parts of silver, and the above process repeated. A trace of rhodium imparts to the bisulphate of soda a yellowish colour, and when this happens the globule of gold suffers a loss of weight compared with the previous weighing.

FIG. 122.—MORTAR.

Gold containing iridium is dissolved in aqua regia, whereby the iridium remains behind as a black powder. This is well washed, and the gold precipitated from the diluted solution with protosulphate of iron.

Gold containing palladium is alloyed with three parts of silver, and treated like an ordinary gold assay, when the palladium as well as the silver goes into solution. After the latter has been removed from the diluted solution by chloride of sodium, the former may be precipitated by metallic zinc.

The Assaying of Gold Quartz.—First of all the quartz is finely pulverised by triturating it in an iron mortar, shown in Fig. 122. This operation is much facilitated by heating the quartz to redness, and then plunging it in cold water. Having pulverised a few thousand grains, it is usual to make at least two assays of the sample to test the correctness of the result. For each of these weigh 500 or 1,000 grains on the pulp scales, Fig. 123, according to the richness of the ore, and well mix on a clean piece of paper with four times the weight of litharge or red lead, and double the weight of carbonate of soda,

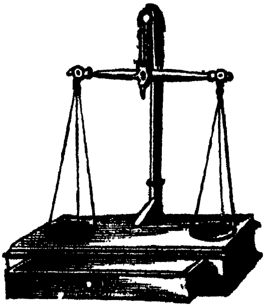


FIG. 123.—PULP SCALES.

and quarter the weight of powdered charcoal. The precise quantities are not very important, but it is better that the carbonate of soda should be in excess than that it should fall short of the proportion given. The following is a very good mixture :—480 grains of the gold quartz, 1,500 grains of litharge, 1,000 grains of carbonate of soda, and 100 grains of powdered charcoal. The mixture is then introduced into an assay clay pot (Fig. 124) and covered with fused and finely pulverised borax, and fixed on a piece of fire-brick in the melting furnace.

CLAY CRUCIBLE.

The mixture above mentioned, when introduced into the crucible, should not fill more than two-thirds of its capacity. This precaution is necessary on account of the effervescence which takes place, in consequence of the displacement of the carbonic acid from the carbonate of soda by the silica, and the combination of the carbon with the oxygen of the litharge. The heat is continued till perfect fusion is effected, and towards the last the temperature is raised to bright redness, until no further effervescence occurs, and a clear liquid homogeneous slag is obtained. The crucible must then be immediately removed from the fire (otherwise the unreduced litharge would be liable to cut through the pot), and as dexterously and rapidly as possible the contents of the pot should be poured into the mould shown in Fig. 125. The mould has the shape of a cone, and on cooling it is turned over, when the slag will be found solidified in the upper portions and the lead button in the bottom. By hammering the mass on a clean anvil the slag will readily detach itself from the lead button, which is now ready for cupellation.

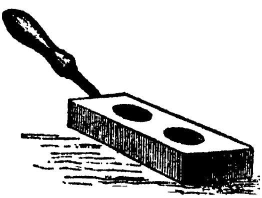


FIG. 125.—MOULD.

The proportion of oxide of lead to be used in the fusion will vary with the amount of oxidizable substances present; but this compound should in all cases be added in excess, since, if the slag retains any traces of an alkaline sulphide, it will retain a part of the gold. Phillips states that, for the assay of iron pyrites, about 30 parts of oxide of lead are necessary, whilst for mispickel, zinc blende, copper pyrites, grey cobalt, and sulphide of antimony, from 15 to 25 times their weight only may be employed.

When auriferous pyrites is the subject of examination, it must first be reduced to fine powder, and then roasted in a shallow dish of refractory clay, heated to low redness in a large muffle, until the odour of burning sulphur ceases to be evolved.

FIG. 126.
ROASTING DISHES.

As the sulphur burns away, the temperature is gradually raised to bright redness. The pyrites is thus converted into oxide of iron. Of this product 1,000 grains or more may be taken and mixed with 500 of dry carbonate of soda, 300 to 500 of litharge or red lead, and 5 per cent. of charcoal, or a corresponding quantity of granulated metallic lead without charcoal, and lastly about 500 of fused borax. This mixture is melted as before, and the fuel is either charcoal, anthracite, or coke.

The principal objection to this method of assay is the large amount of lead which is left for cupellation, since pure iron pyrites requires, when thus treated, 8½ parts of lead, and sulphate of antimony and grey copper ore yield 6 to 7 parts. This inconvenience, as well as the trouble of roasting, may be avoided by the cautious and gradual addition of nitrate of potassa, which effects the partial oxidation of the mineral, and enables the skilful assayer to procure a metallic button of almost any required weight. The nitre, however, if employed in excess, would determine the oxidation of all the metallic and combustible substances contained in the mineral, except the gold, which is never present in sufficient quantity to form a button. The exact amount of nitre to be added must depend on the nature and richness of the ore. As a general rule, 2½ parts of nitre are sufficient to completely oxidize 1 part of iron pyrites; and 1½ and ⅔rds their weight respectively, in the case of sulphide of antimony and galena, are sufficient to produce the same effect on these ores.

Alloys of gold containing tin and zinc must also be fused before cupellation with nitrate of potassa, so as to oxidize these metals; and by adding about 16 parts of lead when the mixture is quite melted, they are separated in the state of slag or scoria, leaving, as before, a button of lead containing the whole of the gold, which is now ready for cupellation.

Assaying by Scorification.—Scorification—or, in other words, the conversion of the silica and other impurities into a scoria—is simply another method of fusion, in which, however, metallic lead, instead of its oxides, is employed; and the oxidation of the various substances to be removed is produced by the aid of atmospheric air, whilst the litharge necessary for the fusion of the earthy and siliceous matters is formed by the oxidation of a portion of the metallic lead. For this operation, instead of a crucible, a shallow cup-like vessel, termed a scorifier, is employed.

The Scorifier.—This vessel is made of refractory clay, and made as compact in structure as possible, in order to resist the corrosive action of melted litharge. The scorifier with its contents is heated in the muffle of an ordinary assay furnace, and as many assays may be introduced at one time as there is room for in the muffle.

Fig. 127 represents, in front elevation and vertical section, the form of furnace at present employed in the Royal Mint, which was arranged by Field, the late assay master, and which is convenient both for this purpose and for the subsequent operation; the muffle used in it is shown at M. This last is an arched oven of fire-clay, the mouth of which is slightly contracted, and it is closed at the back; the closed extremity being pierced with a number of cylindrical holes, in order to insure a free circulation of air through the interior. It is advisable that these holes slope outwards, as shown in the figure, in order to prevent any fuel from falling into the muffle. The top is roughed in order that it may be strengthened by a coating of fire lute. It rests on a plate of fire-clay, beneath which is an iron girder plate, C, placed upon the fire-bars, B. The furnace is made of wrought iron about one-eighth of an inch thick, the whole being lined with fire-bricks. It will be seen that the furnace is provided with five distinct openings: the lower one, A, in order to clean the ash-pit; the two side ones, D D, to remove two fire-bars to admit of the fuel being dropped into the ash-pit; and the two, E and F, to charge the muffle and furnace respectively with cupels and fuel. By means of these perfect control is obtained over the fire, and any required course can be given with facility to the in-going current.

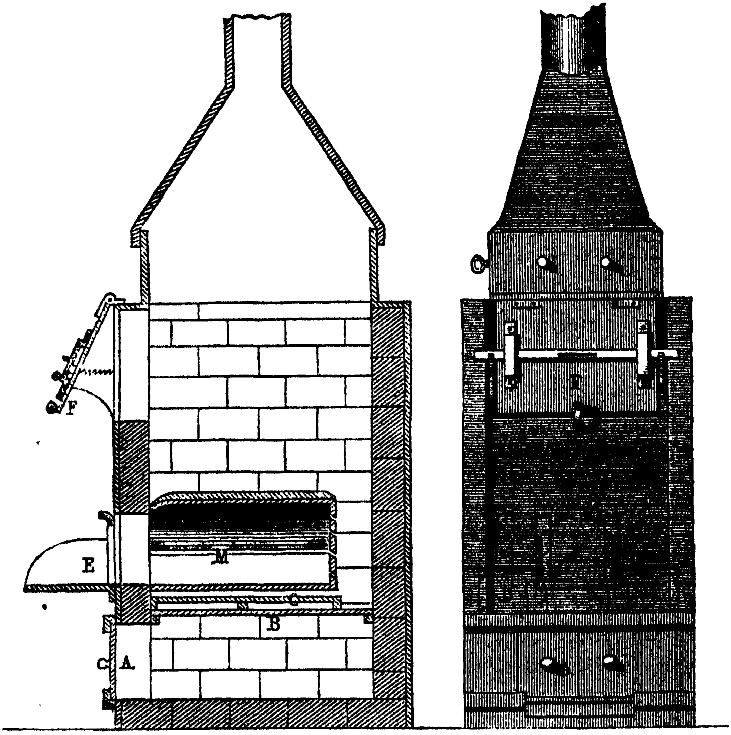


Fig. 127.—Muffle Furnace.

The Battersea muffle furnace (illustrated in Fig. 128) is a portable furnace very suitable for laboratory work. It is built up in sections which are easily packed for transportation. In the illustration *a* is the socket on which to fix iron chimney; *b*, the door for fuel; *c*, the muffle; *d*, the door for muffle; *e*, the muffle arch; *f*, the door for stirring fire; *g*, the door for regulating draft; *h*, the support for muffle, and *i* the iron grate.

Process of Scorification.—Before introducing the ore into the scorifier it is reduced to powder, and a determined weight, about 50 grains, is intimately mixed with from three to five times its weight of granulated lead (the larger quantity being required for poor ores), and one-tenth its weight of borax; this, having been placed in the scorifier (Fig. 129), is covered with from 150 to 200 grains of granulated lead, care being taken that the lead employed contains no silver. The scorifiers charged with this mixture are then placed in the muffle, and the door at the mouth being closed, they are strongly heated for a quarter of an hour. Fig. 130 shows the scorifying tongs, with which the scorifiers are easily handled. By this time the lead will be completely melted, and the

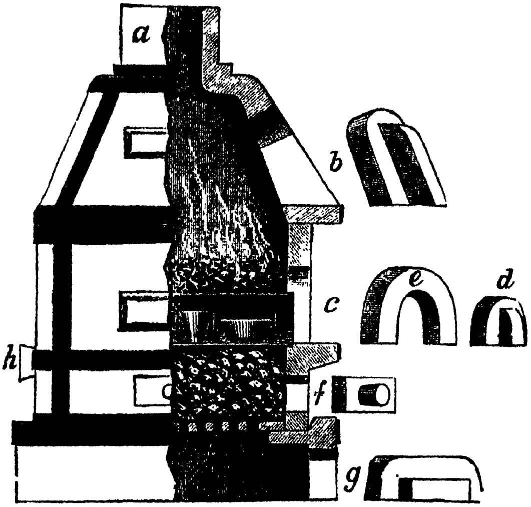


FIG. 128.—BATTERSEA MUFFLE FURNACE.

mouth of the muffle is again opened. Scorification is now effected by the action of the current of heated air passing through the muffle; in other words, the lead is oxidized, and the foreign metals present—whether in combination with sulphur or oxygen—are attacked and dissolved in proportion as this oxide is formed. At first the slag is frequently solid, but it gradually becomes soft, and more and more liquid in proportion as the quantity of the oxide of lead increases.

It is frequently desirable, when the surface has become covered with litharge, to introduce 15 grains of anthracite or charcoal, as by this means a portion of the lead is reduced to the metallic state, and by sinking through the mass of slag takes up any particle of precious metal which may be present. Towards the conclusion of the operation the muffle is for a short time strongly heated, so as to render the slag or scoriæ completely liquid. When a small iron rod, heated to redness and placed in the mixture, is found on being withdrawn to be covered with a slight film of scoria, which runs off clean, the scorification is known to be sufficiently advanced. The scorifiers are successively withdrawn from the muffle by means of tongs, and the contents are rapidly poured into circular ingot moulds of the form represented in Fig. 125. When cold the adherent litharge is detached from the button of lead by a few blows with a hammer. This button contains all the gold and silver which may have been present in the ore, and these are separated from the lead by cupellation.

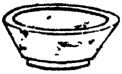


FIG. 129.
SCORIFIER.

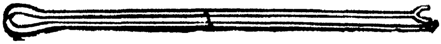


FIG. 130.—SCORIFYING TONGS.

When the material to be scorified is highly carbonaceous, such as the "sweep" or refuse constantly accumulating in the Mint, it is advisable to calcine it previous to the addition of the lead.

If the ore be poor in quality, the whole of the precious metals contained in a large quantity may be concentrated in one small button of lead by repeating the process on successive portions of ore mixed with the requisite amount of lead, and introducing into each assay the button previously obtained. A final button containing all the precious metals from the several quantities of ore will thus be procured. The process may be repeated a third, fourth, or any number of times, till the button of lead thus obtained becomes as rich in the precious metals as may be required. It is necessary, however, that a certain amount of lead be present, with a view to the next process, that of cupellation. In many cases as much as ten times the weight of the ore is employed, but the button may be reduced in size by a single operation to one-sixth or even one-eighth of the original lead. Indeed, it is the chief recommendation of this process, as compared with the method by fusion with litharge, that however small may be the proportion of lead, the slag produced never contains any oxysulphides at the close of the operation, and therefore seldom retains the slightest trace of either gold or silver. The process of scorification has therefore been justly characterised as one of the most exact methods that can be employed, being simple and effective, and applicable, without exception, to the assay of all kinds of auriferous and argentiferous ores.

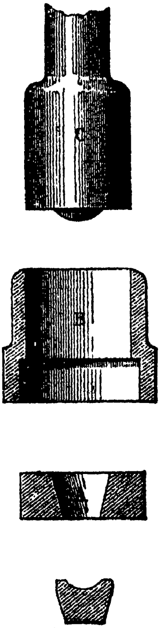


Fig. 131a.

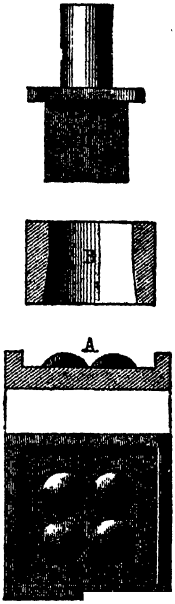


Fig.

Cupel Moulds.

Cupels and Cupellation.—Cupels for assaying are made by pressing moistened bone-earth into a mould of cast iron or gun metal, which varies in construction. Two forms at present in use in the Royal Mint are shown in Figs. 131, A and B, one-half their natural size. In the first of these, after the space in the piece of metal, A, has been rather more than filled with bone-ash, the guide, B, is placed over it. The plunger, C, having been introduced into this guide, the whole is placed under a common lever in the press, by which means the bone-ash is compressed into the form shown in the figure. Formerly the plunger was struck several times with a hammer in order to give the required compression. It is desirable to give a slight twisting motion to the plunger on removing it.

The second form of mould, shown in Fig. 131 B, has been introduced in order to lessen the labour of the furnace manipulation. It will be seen that each cupel is square, and is provided with four depressions. Each of these, however, is rather smaller than the one already described, the amount of metal operated upon in the two cases being about 7·5 to 16 grains respectively. The square cylindrical piece, B, being placed on the plate, A, is filled with bone-ash, and with the plunger, C, is introduced as before into the press, the cupel when formed being five-eighths of an inch deep, square, and slightly tapered at the bottom. The cupels, when formed, should be kept in a dry place for several months, as it is found, when any method of hastily drying them is adopted, that they almost invariably crack when introduced into the muffle and cause the assay to spurt.

Cupellation is performed in a muffle furnace, as described above. At the time of lighting the furnace the requisite cupels are successively introduced into the muffle by light tongs, the floor of the muffle being covered with a thin layer of bone-ash before the introduction of the cupels. The temperature of the muffle having been raised to an orange red heat, the assay pieces—either in the form of buttons mixed with lead, as obtained by scorification, or as samples of alloy wrapped in lead foil, as already described — are successively introduced into the cupels, care being taken that they are so arranged as to

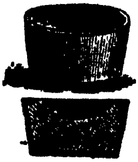


FIG. 132.
CUPELS.

avoid possibility of confusion on their subsequent removal. It is advisable that while the muffle is being thus charged the door should be partially closed by means of a hot fire-brick, in order to check as much as possible the chilling of the cupels which are near to it. The requisite amount of lead may all be introduced as a wrapper of the assay piece. Of course when the alloy for assay consists of a button obtained by a previous process of fusion or scorification, this addition of lead is unnecessary, as a sufficient amount is already contained in the button.

By the previous process of fusion with litharge—or that of scorification, if deemed preferable—the earthy impurities will have been removed in the form of slag or scoria, and nothing now remains but the gold, or a mixture of gold and silver, in combination with the lead and any other metals that may be present. The process of cupellation, as already stated, consists in the oxidation of the lead and other oxidizable metals, and their absorption by the cupel. It is, in fact, a condition essential to the success of the assaying process that the whole of the oxides should be absorbed by the cupel, and thus removed. But this power of absorption varies according to its texture and the care with which the material has been prepared. On an average it may be calculated that it will absorb about its own weight of fused litharge. This, though not strictly correct, will help the experimenter as regards the amount of lead which ought to be mixed with the assay. It is evident, on the one hand, that this must not exceed the absorptive power of the cupel; on the other hand, the actual amount within this limit must depend on the nature of the alloy upon which the cupellation is to be performed.

Weighing the Button of Precious Metal.—The silver button obtained by cupellation of the lead button in an ore assay, if 230 grains of pulp are taken as the assay weight, is freed from the adhering bone-ash by hammering edgeways, and is then weighed. The weight, multiplied by 1·16, gives the amount of ounces per ton of ore of 2,000 lbs., which is illustrated by the following example. For instance, a button is found to weigh—

300 parts
50 „
6 „
356 × 1·16 = 412·9 ounces per ton of ore.

After weighing, the silver button is introduced into a glass tube, adding about half an ounce of pure nitric acid, and heated by the alcohol flame. It soon begins to boil, emits reddish-brown vapours, and leaves the gold, if any in the assay, in undissolved particles of a black colour in the tube. The nitric acid containing the dissolved silver is poured off slowly, and the tube filled with distilled water. When all the particles of gold have settled, this water must be poured off carefully and the tube filled once more with water to the brim.

The tube is then covered with the dry cup and quickly turned over. The gold falls to the bottom of the cup, but being very light and sometimes in minute particles, the tube must be kept for awhile in this position until no suspended particles are visible. It requires some practice to lift the tube without spilling any water, which would invariably carry out some gold. The easiest and surest way is to gradually lift up the tube till the water, the brim of the tube, and the dry cup are level. A slide of the tube, with a dexterous upward movement, leaves the gold in the cup undisturbed. A slight tapping of the cup will bring the gold particles together; the water is then poured off, and the cup dried over the alcohol flame till the gold assumes a yellow colour. The gold is now carefully weighed and calculated upon as follows :—

The gold is found to weigh, for instance, $\frac{35}{1000}$, and the silver button, before dissolving, $\frac{356}{1000}$. If the gold is subtracted from the silver which contained this gold, we thus find the pure silver—

356 − 35 = 321 silver × 1·16 = 372·3 ounces per ton.
and 35 gold × 1·16 = 40·6 „ „

To find the value, the ounces of gold must be multiplied by 20·67, and those of silver by the market value per ounce. Therefore

Silver = 372·3 ounces × $1 = 372·30
Gold = 40·6 ,, × $20·67 = 839·20

Total value $1211·50 per ton.

In assaying gold ores the button will not dissolve in nitric acid. In this case it must be melted after weighing on a cupel with the addition of three times its weight of pure silver, with a little lead foil, and then dissolved and treated as above described.

In case the ore for the assay has been weighed out by half an ounce—equal to 240 grains—the calculation is made in the same way as before, with the exception that the number 1.215 must be substituted for 1·16. The procedure of the preceding example would be as follows :—

The weight of the button was 321. This multiplied by 1·215 will give the amount of ounces per ton of ore of 2,000 lbs.

32 (321) × 1·215 = 390.

The quantity of fluxes used for 250 grains of ore will also serve for half-ounce assays, which can be as follows :—

Ores or Tailings containing but little Sulphurets.

Ore	250 grains.
Glass	125 ,,
Flour	8 ,,
Litharge	1½ ounces.
Soda	1 ounce.

Ores containing about 50 per cent. of Sulphurets.

Ore	250 grains.
Glass	125 ,,
Iron	50 ,,
Litharge	1½ ounces
Soda	1 ounce.

CONCENTRATED PYRITES, NOT ROASTED.

Ore	250 grains.
Litharge	2⅓ ounces.
Soda	350 grains.
Borax	240 ,,
Powdered nitrate of potash	400 ,,
Salt	250 ,,

The salt is placed on top as a covering, and the assay is gradually heated. If iron matte is formed, the assay must be repeated, and the proportion of nitre increased.

CHAPTER X.

CHEMICAL EXAMINATION OF GOLD ORES.

TESTS FOR PRESENCE OF GOLD—Reagents employed—Preparation for Testing—Action of Sulphate of Iron; of Protochloride of Tin; of Oxalic Acid—Tests for Metals associated with Gold: Copper; Silver; Platinum—Application of Reagents—Quantitative Determination of Gold by the Wet Process—Quantitative Estimation by Specific Gravity.

Tests for Presence of Gold.—A rude method of ascertaining the presence of gold in crushed quartz, or earthy ore, is by washing it in a hand-basin or horn spoon; but gold is often present in the matrix in grains or particles so minute that it cannot be detected by the eye, and sometimes it is quite disguised by admixture with other metals or minerals.

In these cases it is necessary to resort to chemical examination. In commencing mining operations the first point to determine is, of course, whether gold exists in any appreciable quantity; and if this be decided affirmatively, the next point is to determine the proportion or quantity of gold contained. The first process is termed testing for gold, or the qualitative examination of a suspected mineral; the second is the quantitative examination, or assaying process, which has just been described.

Sulphate of iron, protochloride of tin, and oxalic acid are the tests or reagents pre-eminently employed in seeking to determine the presence of gold.

Preparations for Testing.—Before applying these tests, the substance supposed to contain gold must be brought into a state of solution; and this can only be done by means of nitric or hydrochloric acid, as already mentioned. If the substance consists of earthy and quartzose matter, this must be reduced to powder by trituration in a mortar before it is subjected to the action of the solvent; but if the matter under examination be simply a metallic alloy, it can be dissolved without any previous preparation, and an excess of acid should be avoided. When the solution is effected the liquid should be evaporated to about one-eighth of its original bulk, and then diluted with 3 or 4 ozs. of water. The action of the reagents is as follows:—

(1.) **Sulphate of Iron or Green Vitriol.**—If a few crystals of this salt be dissolved in distilled water, and dropped into the suspected solution, the result is the precipitation of the gold, if any be present, in the form of a dark brown powder, which is metallic gold in a very fine state of division, as already described in connection with assaying of gold. The deposit formed when the sulphate of iron is added may be corroboratively proved to be gold by its being insoluble in nitric acid, but readily soluble in aqua regia.

(2.) **Protochloride of Tin.**—If to another portion of the nitric hydrochloric solution be added a small quantity of a solution of protochloride of tin (commonly known as salts of tin), there will be immediately produced, if any gold is present, a dark brownish-purple precipitate, known as purple of cassius. This substance is used in enamel and porcelain painting, and for giving to glass a tinge of a fine red colour. Its colour, though not a brilliant purple, but rather a reddish-brown, is characteristic, and after being once seen is not likely to be mistaken. Its appearance, when the chloride of tin is added to the liquid, affords an infallible proof of the presence of gold, for a very minute portion of that metal gives a manifest reaction when this test is employed.

When the first test (sulphate of iron) has been applied, and its evidence attested by the solution of the precipitated gold powder in aqua regia, the protochloride of tin may be employed to produce the purple of cassius in this solution also.

(3.) **Oxalic Acid.**—This substance, either in crystals or dissolved in water, causes, when added to the solution of gold, the precipitation of any of that metal in a form which varies with the temperature and strength of the solution.

The action takes places very slowly in the cold, but on applying heat there is immediately a copious evolution of carbonic acid gas, while hydrochloric acid is also formed during the process.

A crystal of oxalic acid, wetted with a solution of gold, becomes soon covered with a thin film of the metal. There are also other reagents which act on gold.

Tests for Metals usually associated with Gold.—In examining an ore or alloy supposed to contain gold, it is often an object of great importance to determine the nature of the metals with which the gold is associated. These may be of high value on their own account, and, according to the quantities in which they are present, may greatly affect the value of the ore or mineral in question. To enter fully into this subject in connection with the rarer metals that are found associated with gold would be of little practical use. Attention should, however, be especially directed to the metals of common occurrence—namely, copper, silver, and platinum.

(1.) **Copper.**—It has been shown that copper is almost always associated with gold, even in quartz, and that copper pyrites is one of the substances frequently mistaken for gold. When dissolved in acids, however, it gives characteristic reactions which render its presence easily distinguishable. One of the readiest tests is to introduce into the solution a piece of clean iron, when, if copper be present, it will be deposited on the iron in the metallic state, an effect which is due to a simple exchange of place between the metals, the iron passing into solution while the copper exhibits itself in the solid form.

Again, when ammonia is added in excess to a solution in which copper exists, it communicates to the liquid a rich deep blue colour. Ferrocyanide of potassium produces with copper a brownish red precipitate, even when the metal is present in very small quantity. Carbonate of soda precipitates copper from its hot solutions in the form of an apple-green compound, which is a carbonate of copper known, when artificially formed, as *verditer*, and when it occurs native, as *malachite*. Copper ore in the latter form exists abundantly in Australia—not, indeed, mixed with the gold, but constituting valuable mines, from which the ore is sent over to this country to be smelted in South Wales.

If a platinum wire be dipped in a copper salt solution containing free hydrochloric acid, and then held in the flame of a Bunsen burner, the flame will show an emerald green tint. This test is applicable to the detection of exceedingly minute quantities.

(2.) **Silver.**—It has been shown that gold is invariably alloyed with this metal, sometimes to a very large amount. In its separate state it is readily distinguished, not only by its white colour, but also by its specific gravity, which is only 10·4, or about one-half that of gold. It may be easily recognised by its chemical behaviour with reagents, in which respect it differs from gold by its solubility in nitric acid at all temperatures, and in hot sulphuric acid. On the contrary, with hydrochloric acid it forms a white curdy precipitate, which is the chloride of silver. If the nitric acid employed to dissolve it contains the least hydrochloric acid, the solution will become turbid by the formation of the chloride. Hence, when a mineral containing gold and silver is submitted to the action of aqua regia, the appearance of this white precipitate will immediately indicate the presence of the latter metal. The chloride of silver is soluble in ammonia, and may thus be distinguished from many other white precipitates; or it may be further tested by putting the precipitate into a crucible with carbonate of soda, and exposing the mixture to a strong red heat, when a button of pure silver will be obtained. By careful manipulation the amount of silver present may, in this manner, be accurately determined. If the mineral containing the silver be dissolved in oil of vitriol, the metal is readily detected by inserting a few fragments of copper, which will cause the precipitation of the silver in a pulverulent state.

(3.) **Platinum.**—This is another metal frequently associated with gold, and as it is one of the noble metals, and ranks in price between silver and gold, an ore which contains it in any quantity is of considerable value. The specific gravity of platinum is about 21·50—higher, therefore, than that of gold; indeed platinum, osmium, and iridium are the heaviest of all known metals, all three possessing about the same specific gravity; possibly osmium is slightly heavier than the other two. Platinum is of a light, steel-grey colour, less ductile than gold or silver, but more tenacious; indeed it will support greater weights on equal thicknesses of wire than any metal excepting iron and copper. It is distinguished from gold not only by its colour, but also by its extreme difficulty of fusion; it does not melt by itself in the highest heat of a furnace, but softens sufficiently to admit of forging and welding, and in the arc of flame of the voltaic current, or before the oxyhydrogen blowpipe, it admits of being fused even in considerable masses. On the other hand, it resembles gold not only in its high specific gravity, as already stated, but also in the fact that it resists the action of the simple acids, and is soluble only in aqua regia. This circumstance, together with its great infusibility, renders it of important use in many of the arts, and indispensable for various purposes in the laboratory. It is the metal universally employed for apparatus which require to be exposed to high temperature and powerful chemical agencies, without undergoing any change. It is, therefore, very important that miners in the gold fields should look carefully for this metal, which, being destitute of any glittering appearance, is likely to escape the notice of an ordinary observer.

There are, however, certain chemical reactions by which platinum may be readily distinguished and separated from gold in a solution. Sulphate of iron and oxalic acid, which precipitate gold, do not precipitate platinum. When the latter is dissolved in aqua regia, and the acid neutralised by carbonate of soda, the metal is deposited as a black powder, if the mixture be boiled with tartaric acid and soda. Further, the addition of chloride of ammonium and alcohol to a strong solution of platinum causes the deposition of a yellow crystalline precipitate, which is characteristic of this metal.

Application of the Reagents.—Having thus described the properties and characteristic reactions of the different metals which it is desirable to look for as being frequently associated with gold, let it now be assumed that the substance to be examined is a piece of auriferous quartz. This must be first reduced to powder, and then boiled for some time in an earthenware or glass dish with aqua regia. The solution is then diluted with water, passed through a filter, and allowed to cool. If any silver be present it will remain in the filter as a white precipitate mixed with the quartz.

To the liquid which has passed through the filter a solution of carbonate of soda is now added until no more effervescence takes place. This will precipitate all other metals which may be present except gold and platinum, which will remain in solution. The liquid is again filtered, and a solution of oxalic acid added until it ceases to produce effervescence and has a sour taste; then boil, and if gold be present it will be precipitated as a dark powder. The platinum, if any be present, will still remain in solution.

Decant or filter the liquid from the gold precipitate, and add to the former protochloride of tin, when a reddish brown colouring will appear if platinum be present. Or by boiling with tartaric acid and soda, the platinum will be thrown down as a black precipitate.

It has been stated that if silver be present it will be found on the first filtering mixed with the quartz. Wash this with ammonia, which, if copper be present, will produce a deep blue tinge. To the solution which comes through add hydrochloric acid until the smell of ammonia disappears, and the silver will be thrown down as a white curdy precipitate.

It is evident that other methods and reagents may b adopted. For example, the original solution in aqua regi may be concentrated by evaporation, until it is very muc reduced in quantity; then add about three-fourths of its bulk f spirits of wine, and lastly a saturated solution of chloride f ammonium. By these reagents the platinum will be thrown own as a yellow crystalline precipitate, while the solution ltered from this, and treated with sulphate of iron or boiled ith oxalic acid, deposits gold.

By carefully weighing the gold obtained, the amount present in a given quantity of the ore or alloy may be exactly determined.

Quantitative Determination of Gold by the Wet Process.—I will here give the method as indicated by Fresenius in his "Quantitative Analysis."

The quantity to be operated upon should be small, 12½ grains being a very convenient quantity, as it is only necessary to multiply the result by eight to obtain the percentage composition. If the gold be mixed with quartz or earthy matter, so much of this should be taken as may be judged from preliminary experiments to contain the amount of native gold above mentioned; and this must be triturated in a mortar with great care before subjecting it to the action of the aqua regia, the quartz has to be reduced to a state of minute division, to liberate all the gold if possible, and not to leave it oped with silica, which would prevent the action of the acid.

Fresenius recommends that the substance under examination, having been weighed, be warmed with hydrochloric acid, nitric acid being gradually added until solution is effected, or it may be repeatedly digested with strong chlorine water. This latter method is especially applicable when the gold is present in very minute quantities, or mixed with other oxides which it is wished to leave undissolved.

The gold is always weighed as a metal, and its compounds be brought into this state (*a*) by ignition, (*b*) by precipitation as metallic gold, (*c*) by ignition preceded by precipitation as tersulphide of gold.

(*a*) This method is applicable to all compounds of gold which do not contain a fixed acid. The compound should be heated in a covered porcelain or platinum crucible very gently at first, but finally to redness; the residue is pure gold. These results are the most accurate.

(*b*) In the method of precipitation as metallic gold, two cases may occur which may conveniently be considered separately, namely: (1) That in which the solution is free from nitric acid; (2) that in which it contains nitric acid.

(1.) If the solution does not already contain free hydrochloric acid, add a little; then add a clear solution of sulphate of protoxide of iron in excess; heat gently for a few hours until the precipitated fine gold has completely subsided; filter, wash with hydrochloric acid first, and afterwards with water; dry, and ignite. It will be found preferable to form the deposition in a porcelain evaporating dish rather than in a beaker.

(2.) When nitric acid exists in the solution, it should be evaporated on a water-bath to the consistence of syrup, adding from time to time hydrochloric acid. Dissolve this residue in water containing hydrochloric acid, and proceed as directed in (1). It should be noticed that cases may occur in which the residue does not dissolve to a clear fluid, but this is of no consequence.

(*c*) In the case in which ignition is preceded by deposition as a tersulphide, a stream of sulphuretted hydrogen should be transmitted in excess through a dilute solution of the compound under examination. The precipitate thus formed is speedily filtered off without heating, washed, dried, and ignited, as directed in *a*.

In the method *b*, the liquid should be tested before pouring off, to make sure of the last traces of gold having been deposited. This may be done in the following manner: when the whole of the precipitate appears to have settled to the bottom, a few drops of the supernatant liquor are taken out on the end of a rod, placed upon a surface of clean white porcelain, and tested with a drop of protochloride of tin. If no purple precipitate be formed, it is a proof that the whole of the gold has been thrown down. If a dark brown colouring, but still no subsidence, be produced, this will indicate the presence of platinum. If any precipitate be observed, more of the sulphate of iron or oxalic acid must be added to effect a complete precipitation of the gold contained in the solution.

The pure gold after ignition is placed in a watch-glass or small capsule, and accurately weighed in a pair of delicate scales, which should be capable of turning with a difference of at least a hundredth part of a grain. Supposing the gold to weigh 10·75 grains, and that of the amount of alloy submitted to experiment was 12·5 grains, or the eighth part of one hundred, it is evident that $10·75 \times 8 = 86$ is the percentage of pure gold present.

Quantitative Estimation by Specific Gravity.—The high specific gravity of gold permits of its being estimated approximately in an ore or mineral of known specific gravity. The additional weight must be due to the presence of gold, and hence the amount of the latter may be estimated.

Supposing we have a rich specimen of gold quartz, and wish to know how much gold the same contains, we proceed as follows: we take the weight of the piece in the air, then suspend it by a string from the scales and let it plunge in a tumbler of water and note its weight. The principle, therefore, is based on the well-known theorem of Archimedes: "When a solid is immersed in a fluid it loses a portion of its weight, and this portion is equal to the weight of the fluid which it displaces; that is, to the weight of its own bulk of that fluid." The weight of the piece of quartz when weighed in water will be less; the lessened weight being determined, nothing more is required than to find by division how many times the latter is contained in the former, and the quotient will give the density or specific gravity of the sample under examination. If we find that this is no higher than 2·6, it will be unnecessary to proceed further; but if higher, the amount of gold present may be calculated from the difference.

Let A represent the weight of sample in air in ounces; let B represent the weight of sample in water in ounces; let C represent the specific gravity of gold; let D represent the specific gravity of the quartz; let X represent the weight of gold in sample.

Then A less X represents the weight of the quartz;

$\frac{X}{C}$ represents the weight of water displaced by the gold in the sample when weighed in water;

$\frac{A-X}{D}$ represents the weight of water displaced by the quartz in the sample when weighed in water;

$\frac{X}{C} + \frac{A-X}{D} = A - B =$ the total water displaced by the sample; or, to put the equation in another form—

$$X = \frac{— DCB — AC}{D — C}$$

Knowing the numerical values of A, B, C, and D, substitute them in the equation; then multiply, subtract, and divide as indicated to find the value of X. By applying this method to a sample of gold quartz containing no impurities, such as pyrites, &c., the approximate amount of gold in the specimen can be determined without crushing. The specific gravity of gold can be taken as 19·2 and of quartz as 2·6.

Let us suppose—

A = 26 ounces, B = 22 ounces;

$$X = \frac{(2{\cdot}6 \times 19{\cdot}2 \times 26) — (2{\cdot}6 \times 19{\cdot}2 \times 22) — (26 \times 19{\cdot}2}{2{\cdot}6 — 19{\cdot}2};$$

C = 19·2 specific gravity of gold; X = 18·5 ounces of gold;

D = 2·6 specific gravity of quartz;

$$Q = \frac{DC\,(A - B) - D \times 2{\cdot}6}{19{\cdot}2 — 2{\cdot}6};$$

X = weight of gold

Q = weight of quartz

$$Q = \frac{2{\cdot}6 \times 19{\cdot}2\,(26 — 22) — 24 \times 2.6}{19{\cdot}2 — 2{\cdot}6} = Q = 7{\cdot}95 \text{ ounces}$$

of quartz.

CHAPTER XI.

PHYSICAL AND GEOLOGICAL FEATURES OF GOLD-PRODUCING COUNTRIES.

THE SIERRA NEVADAS—The "Mother Lode" of California—Hot Springs of California and Nevada.—Other Features of those Countries—Death Valley—Hot Mud Springs—Telluride Ore Veins of Colerado—Origin and Characteristics of Gold Deposits—Placer Mines of California—Gold in Australia—Chemical Geology of Gold.

The Sierra Nevadas.—As this mountain range of California—a region 300 miles long by 50 miles broad—owes its celebrity to the enormous amount of precious metals which have been produced from its numerous ravines, gullies, rivers, and quartz lodes, some observations here upon the geology of its mountain chains, and the distribution and probable origin of the precious metals, may prove of interest to the reader.

There is probably no country on the face of the earth which in respect of natural features offers to the student so many points of attraction as California, while for scenic effects it has no equal.

Among the numerous quartz lodes, the most prominent are those found near the line of separation of the granites and the schistose rocks. One great quartz lode—the "Mother Lode" of California—having a thickness of from 2 to 20 yards, can be traced for about sixty miles, cropping out in some places like an immense wall. Professor Whitney, State Geologist of California, says of this vein :—

"It is by no means a continuous bed or vein of quartz, but rather a series of nearly parallel belts or lenticular masses, with barren intervals between them, but yet arranged nearly in the same course, so that a straight line drawn in a direction of north 27° west from Jackson, in Amadar county, would either cut or pass through very nearly the whole of them.

"Yet this lode can be traced, by a succession of massive croppings and other easily recognised tokens of its presence, along a nearly straight line for a distance of about sixty miles. Throughout its entire length—from the Princeton and Josephine mines, on the Mariposa estate, to the Keystone mine, near Drytown, Amadar county—there occur at intervals vast wall-like outcrops of quartz, forming an impressive and picturesque feature of the landscape. The vein, in its upper portion, pitches toward the east at an angle of 45° to 50°, gradually approaching a vertical position as depth is attained.

"In thickness or width it varies from 1 foot to 50 feet of solid quartz, but sometimes expands to a width of 100 feet or more, in which case the vein matter is partially filled with fragmentary slate and broken quartz. It is everywhere regularly walled, and uniformly carries a well-defined 'gouge' or selvage of argillaceous clay, greatly facilitating the extraction of its ores. The western or foot wall is slate or greenstone, and the eastern or hanging wall, metamorphic slate, sometimes termed 'bastard granite' by the miners.

"The vein matter of the mother lode consists of grey or white quartz, impregnated with about 1 per cent. sulphurets, and rarely producing 'specimen rock' or free gold-bearing quartz. Generally the gold is disseminated throughout the vein stone in such minute particles as to be invisible to the naked eye.

"The ore does not occur in a continuous channel, but in a series of chimneys or pay shoots, separated by stretches of low grade ore, barren quartz, or argillaceous slate. The longitudinal inclination of these chimneys is sometimes northward, and at others southward. In vertical continuity none of these pay shoots have, up to the present time, been completely cut off; and the experience of over thirty-five years' exploration on this lode has demonstrated the fact that the pay shoots may continue to a depth where mining, unless there should be a great improvement in the grade of the ores, will cease to be profitable.

"The Sierra Nevada itself consists mainly of a central axis of granite, disturbed at intervals by volcanic action, in which is found only a small quantity of quartz. Along the western slope of this mass are thick bands of slates and shales, alternating with masses of serpentinous and trap rocks."

The slates of this locality belong (according to the same authority) to the Jurassic period, although in all probability some of those in Plumas county and elsewhere are of the Triassic. The great metallic belt lies on this western slope, commencing at about the Tegon pass, and extending to the northern limit of the state.

Judging from the immense quantities of quartz débris which are accumulated in the auriferous gravels of the country, it is certain that quartz accumulations must have formerly been in greater abundance and covered a larger area than would be inferred from the appearance of the existing veins of to-day.

In regard to mineral wealth, California occupies a prominent position, not only with reference to the quantities of the precious metals, but also to those metallic elements which occur but sparingly in other countries. Besides gold and silver, platinum, mercury, copper, lead, antimony, arsenic, tungsten, tellurium, molybdenum, bismuth, chromium, manganese, iron, nickel, cobalt, and zinc, are found in various ores.

If we look at the fissure veins on one hand, and observe on the other the immense masses of erupted rocks in these mountains, one cannot fail to suspect a connection and ascribe the production of fissures to volcanic forces, or to earthquakes that accompanied the eruption of molten rocks. But how, may we ask, were these cracks filled with vein matter? Neither the lateral infiltration hypothesis nor the injection hypothesis is sufficient to account for all peculiarities met here; but if we consider with Elie de Beaumont and H. Daubree that veins are the product of hot waters which, charged with mineral salts, entered these immense fissures from beneath and filled them to the brim, we may easily explain the presence of the quartz, of carbonate of lime, and of carbonate of lead and copper, as there is not a single metallic carbonate that would not be a little soluble in water charged with free carbonic acid.

The occasional banded structure of the veins is also thereby explained, the deposits being made at first upon the walls of the fissures, and gradually filling up to the centre. If the character of the mineral water changed after a certain period, a change of the nature of the deposit would of course be the result, and thus the banded structure of various ores—which also is seen occasionally in veins—would be accounted for. The deposition of vein matter from the hot waters is partially due to a loss of temperature of the water, partly to a loss of carbonic acid in contact with the air at the fissure surface. The formation of the metallic sulphurets might be explained by the subsequent entering of waters charged with sulphuretted hydrogen, converting the metallic carbonates into the sulphurets or sulphides.

No other country abounds so largely with hot springs as do California and Nevada, and it is especially in the vicinity of mining districts that they are encountered. Who would doubt that the system of thermal springs was formerly much more extensive in California than at present, when one sees the glaring and decisive marks they have left, the cones they have built, the coatings produced? A fine specimen of a snow-white coating over palæozoic limestone is seen on the eastern slope of the Buena Vista mountains, three miles east of Cerro Gardo, Inyo county.

On subjecting the surface conditions of California to a critical examination, one cannot fail to be struck by the great number of thermal springs. Indeed, there are few countries in the world with so large a number within an equal area. It is true, Montana and Idaho, with the famous geyser regions, contain also a considerable number; and Nevada still more. But these territories, more or less, resemble California in regard to volcanic formations. There is no doubt that a close connection exists between the latter and the thermal springs, both being due to the fact that the earth crust in this region covering the molten interior is thinner than elsewhere. Waters, after penetrating this crust, become heated, charged with salts, and are driven through other fissures to the surface by the power of the generated steam; and it is to this relatively thin crust that we may look for an explanation of the numerous earthquakes occurring in California.

Alexander von Humboldt, in describing his travels through the equatorial regions of South America, says that "the hypothesis of the relation between the volcanic formations and the existence of thermal springs seem *not* to be well founded," simply because he encountered springs of nearly boiling temperature issuing from Azoic rocks at Mariara on the Orinoco. He certainly would have formed a different opinion had he travelled through New Mexico, Nevada, and California, where gigantic peaks and wide-spread flows of volcanic material form most prominent features. It may safely be assumed that it is the exception for thermal springs to issue within very large distances from volcanic formations.

A fact of no little interest is the occurrence of a number of lakes in Eastern California containing as their main constituent carbonate of soda. These lakes, and a number of alkaline and saline flats, are situated east of the Sierra Nevada, and west of the great parallel ranges known as the White Mountains, Inyo and Argus ranges, which are filled with splendid fissure veins. The alkaline lakes are Owens Lake, Mono Lake, Black Lake, and the slough of Bishop's Creek.

In Lower California, as in Eastern and Central Nevada, and in Utah, we find vast depositions of salt, saline efflorescences, and alkaline deposits as well as salt-water lakes. A striking feature is Saline Valley, in Inyo county. If one stands upon the summit of the Beveridge range of the eastern slope, and looks towards the east—but down, down until the senses swim with the awful depth and distance—Saline Valley is seen stretching along the base of these mountains, a great basin, a desert valley, rimmed by volcanic hills, its wastes of billowy sand, its sloping dry washes of rock and boulder, its shining patches of soda and salt blazing under the

the quartz, of carbonate of lime, and of carbonate of lead and copper, as there is not a single metallic carbonate that would not be a little soluble in water charged with free carbonic acid.

The occasional banded structure of the veins is also thereby explained, the deposits being made at first upon the walls of the fissures, and gradually filling up to the centre. If the character of the mineral water changed after a certain period, a change of the nature of the deposit would of course be the result, and thus the banded structure of various ores—which also is seen occasionally in veins—would be accounted for. The deposition of vein matter from the hot waters is partially due to a loss of temperature of the water, partly to a loss of carbonic acid in contact with the air at the fissure surface. The formation of the metallic sulphurets might be explained by the subsequent entering of waters charged with sulphuretted hydrogen, converting the metallic carbonates into the sulphurets or sulphides.

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The dryness of the air is so excessive that moisture is withdrawn from the body faster than it can be supplied through the system. From this cause frequent cases of death have occurred even when water was abundant, as it could not be drunk fast enough to supply the drain caused by the dessicative power of the dry hot air. I have had a similar experience in a neighbouring region; and I have known also instances of prospectors setting out to explore the Coso Ranges, who in crossing Panamint Valley have been seized with a fever, followed by delirium. Becoming bewildered, they have wandered about aimlessly, beholding in the far distance lovely images of green vegetation, brooks of clear water, and with blazing eyes and panting breath they have hurried on to reach the spot, even divesting themselves in their despair of every particle of clothing, till under the burning heat of the desert they have fallen in death.

Little vegetation can be found a short distance from water, excepting sage brush. Near the creeks only grass, willow, and mesquit brushes grow. The heat is intense, averaging from 95° to 105° Fahrenheit in the shade, and I have noticed it at the Modoc mine 126° in the month of July. Water can be found in abundance by sinking wells in almost any part of the valley.

The Indians mark the site of springs by placing large white stones at the mouth of each cañon leading from the valley in which there is water; and on looking up the cañon, other stones are seen similarly placed, which lead to water if followed. Elsewhere it is useless to look for natural springs.

Hot Mud Springs.—Near Coso District are hundreds of these springs, of great interest, some in constant motion, boiling and bubbling up mud. There is one oblong basin, 150 feet long by 75 feet wide, filled with clear alum water, which ebbs and flows every few minutes. It gradually rises from 4 to 5 feet, and as slowly sinks again. A white rock thrown into the spring can be seen to sink for a minute or more. The ground round about is hot. Half a mile west lie extensive banks of sulphur. From crevices steam issues, and on the rocks sulphur in beautiful crystals sublimes. Similar mud springs are found in the Colorado Desert. Four miles distant are low hills of obsidian, several extinct volcanic craters, and walls of lava and pumice-stone, all showing that volcanic agencies were once very active at this locality.

Telluride Ore Veins in Boulder County, Colorado. —This county is remarkable for its valuable belt of telluride ores, carrying gold and silver, which is found in the southern part of the county, the formation being three miles in width by about thirteen in length. Only twice before in the history of mining had telluride ores been encountered in any quantity —in the mountains of Transylvania, and in Calaveras county, California—but in neither instance did they appear in such novelty and abundance as in Boulder County.

The telluride belt, as at present known, extends in a northerly direction through Gold Hill, which is five miles from its southerly extremity. The country rock, which at the northern part of the belt is micaceous or gneissic schist, is in the remaining portion principally gneissic granite. All the lodes carry mineral of a similar character; but they may be divided into three classes, each distinguished by the kind of rock inclosing it. First, lodes in gneissic granite, embracing the richest and largest number of mines; second, lodes between granite and porphyry contacts; third, lodes in micaceous or gneissic schist, including the remaining veins, among which is the Smuggler, probably the largest telluride vein yet opened in the country.

Mr. John K. Hallowell, in a published monograph of Boulder county, gives the following description of the geological structure of the rocks of the district:—

"At a time when the system of elevations known as the Rocky Mountains had no existence, the space now occupied by them was an ocean of vast extent, but dotted here and there by islands of primitive granite, the earliest or oldest rock known. Around these islands sedimentary rocks were being formed, which were of various thicknesses and extent, and which afterwards, by pressure and heat, became metamorphosed. Such was the condition of the granite mountains Gold Hill and Big Horn, just east of it, in the olden age. Boulder City to-day is but ten or twelve miles east of them. This granite island shoaled off from twenty or twenty-five miles south. Both on the east and west sides of this primitive formation heavy beds of sedimentary rocks (afterwards metamorphic) were deposited, gradually filling the ocean bed to a level, when on the southern part of this island they overlaid it many feet in thickness. But compared to the thickness of the strata built upon the sides of the granite, the overlying part was a mere shell. The metamorphic rock on the east side only came as far north as Boulder Creek. North of that, and east of the mountains named, there was none. As this ocean gradually shallowed, on the metamorphic rocks was deposited the sediment that afterwards made the cretaceous rocks of the plains and foothills. At this place or island on the east side, a few hundred feet only of this age of sedimentary rocks were laid directly upon primitive granite. The other formations are entirely wanting. At this time we have a shallow, marshy sea, with granite islands showing a small distance above the water-level. Abundance of vegetation in surrounding marshes, which afterwards made the lignite coal beds of the cretaceous measures. That this condition of things continued for a long time, and with some slight oscillations of level, is evident by the different beds and quantity of the coals. Finally came the contraction and wrinkling of the earth's crust, the result of which was the Rocky Mountain system, which, as it arose from the ocean depths, brought up as dry land the whole system of the plains from just west of the Missouri River, as well as the adjoining lands through to the Pacific Ocean.

"The force that elevated the mountain range carried the granite islands up also. In the movement large fissures opened in the granite, miles in length, and parallel with the longest axis of the island. The metamorphic rocks broke at their thinnest place, viz. just over the granite, and stand to-day uptilted upon each side of it. Those strata to the east dip towards the plains, those west towards the range proper.

"The different dips of these metamorphic rocks puzzled me for over a year, as when working in Clear Creek County in 1880 I noticed it, and could get at no legitimate cause, although in South Clear Creek, a short distance above the forks, I could see where the strata broke; and at this place, on the north side of the creek, I found primitive granite overtopping with knife-blade streaks of tellurium ores. At that time I could not understand all of its significance, until a study of the foothills at Boulder City led me to connect the two points, and so solve the problem of the structure. At Clear Creek this split of the metamorphic rocks turned west a couple of miles, then south-west several miles. The bed of South Clear Creek occupies the chasm to the mouth of Chicago Creek, and this creek follows the rent in the metamorphic rocks almost to its source. Now this split and fissured granite held water like a sponge, the motion of the rocks gave great heat, the heated waters sought the surface through these fissures, at the surface the thermal waters rapidly evaporated, leaving their solids in the fissures. Thus were formed the veins of this tellurium belt.

"Here occurs a beautiful illustration on a stupendous scale of the theory of mineralogists, that the contents of veins are derived from atoms carried in aqueous solutions from the surrounding country rocks—that the contents of the vein are thereby governed by the character of the country rock. Here in the primitive granite exclusively do we get the veins carrying the tellurium ores and the sulphides contained in the other veins. In the parallel metamorphic rocks between the tellurium belt and the range we get the veins with the ordinary sulphides, but no tellurium.

"In one place only are tellurides known where the outcrop is in recent formations—that is, in the Bassick Mine of Custer County, Colorado. To my mind this indicates that the source of the ore supply extends down through all the overlying rock, to a fissure in the underlying primitive rock.

Consequently the deposit of ore in the Bassick Mine will be found to be inexhaustible by any means known at this date that man can work with.

"There are more of these tellurium belts in Colorado, some of them, perhaps, as extensive and rich as the described veins of Boulder County, but nowhere so accessible and well proven."

Origin and Position of the Gold Deposits.—Auriferous rocks, in whatever part of the world they be found, almost always exhibit a striking resemblance to each other; and although at times gold is discovered with granite and other rocks as a matrix, it is generally associated with quartz. These rocks are of the Silurian period, and it has been observed that the richest deposits are found where rocks of this age have been disturbed by volcanic action. "The most usual original position of gold" (says Sir R. Murchison) "is in the quartz ore veinstones that traverse altered palæozoic slates, frequently near their junction with eruptive rocks. Sometimes, however, it is also shown to be diffused through the body of such rocks, whether of igneous or of aqueous origin. The stratified rocks of the highest antiquity, such as the oldest gneiss or quartz rocks, have very seldom borne gold; but the sedimentary accumulations which followed, or the Silurian, Devonian, and Carboniferous (particularly the first of the three), having been the deposits which, in the tracts where they have undergone a metamorphosis or change of structure by the influence of igneous agency or other causes, have been the chief sources whence gold has been derived."

It is noteworthy that the great gold-fields of Australia extend without interruption along the slopes of the range separating the eastern seaboard from the Continent, while the gold-bearing mountains of California and British Columbia extend in an unbroken line along the western shore of America. Indeed, the great American Cordilleras, which is auriferous and metal-bearing throughout its whole length, extends in an unbroken line, from Behring Straits in the north to Cape Horn in the south, bordering the coast of the Pacific Ocean along the North and South American Continents. The Australian range extends southwards across Bass's Straits, and northwards it crops out at intervals in the various islands, finally terminating in Japan. Thus the great basin of the Pacific has on each side an elevation of volcanic origin, from which at intervals gold is obtained.

Characteristics of the Gold Deposits.—The chief sources from which gold is obtained, are, firstly, alluvial deposits; and, secondly, quartz rock—commonly a vein of greater or less thickness, intersecting beds of granite or layers of slate. The former, consisting of sand and gravel, have been produced by the disintegration of the siliceous, granite, and other igneous and metamorphic rocks, and have been transported by the agency of water from the mountainous districts. In these beds it is invariably found in the metallic state, but is never quite pure. It usually contains a certain proportion of silver, and not unfrequently iron, copper, and small quantities of several other metals.

In the Californian gold region the gangue of the auriferous veins is invariably quartz, which is generally crystalline in structure, though sometimes partially vitreous. The veins are irregularly distributed among the metamorphic slates, but occur chiefly in the neighbourhood of crystalline and eruptive rocks.

The quartz of a veinstone is generally ribboned, so as to form a succession of layers parallel to the walls of the lode, and frequently some one or more of these laminæ are more productive than the rest. The metallic minerals, ordinary iron pyrites, blende and galena, arsenical pyrites, copper pyrites, and some silver ores, invariably accompany the gold.

"How gold came to be present in quartz veins" (says Professor Whitney) "is very hypothetical. All that can be said is, that it was evidently deposited from the solutions which gave rise to the formation of the inclosing quartz; and this appears the more evident from the presence of that metal in pyrites inclosed in siliceous incrustations, as well as from the fact of large quantities of gold having been formed in the interior of the stems of trees, which in deep diggings are often converted into pyrites.

"The constant presence of iron pyrites in auriferous veins, and when so occurring its invariably containing a certain amount of gold, suggests the possibility of this sulphide being in some way necessarily connected with the solvent by which the precious metal was held in solution. It has been shown that finely divided gold is soluble in the sesquichloride of iron, and more sparingly in the sesquisulphate of that metal. It is also well known that iron pyrites sometimes results from the action of reducing agents of the sulphate of that metal. If, therefore, sulphate of iron, in a solution containing gold, should become transformed by the action of a reducing agent into pyrites, the gold, at the same time being reduced to the metallic state, would probably be found inclosed in the resulting crystals of that mineral."

The Placer Mines of California.—In the early days of Californian history all the gold was obtained from the alluvial deposits, and after these became exhausted and the deep gravel mines were attacked, attention was drawn to the quartz mines. The "shallow placers" were formed at a more recent period than the "deep placers," which were deposited by an entirely different river system from that now in existence. These ancient rivers evidently flowed at a much higher level, and frequently at right angles to the valleys of the present day. The beds of these rivers—which bear the name of the ancient Californian Pliocene rivers—belong to the Tertiary age, and are most likely of the later Pliocene epoch. Their apparently unlimited deposits of metalliferous gravels are now worked on a gigantic scale by what is known as the "hydraulic process," of which an account has been given in the first chapter of this work.

Formation of the Gold Deposits.—In further elucidation of this question, it will be well to give the views of Professor B. Silliman.* It is susceptible of proof (he says) from numerous well-established facts that at the close of the geological epoch, just prior to the appearance of man upon the earth, the whole of the western slopes of the Sierra Nevada Mountains were, below a certain horizon, covered by a vast spread of alluvium, owing its origin, probably, to the action of extensive glaciers, which have left the evidence of their former presence everywhere in the higher Sierra. The glaciers furnished the transporting power that brought from above the fragments which, by long-continued action of running water, were worn into the smoothly rounded boulders, gravel, and sands forming the gold-bearing alluviums.

The melting of the glaciers as their lower skirts reached the warmer zones furnished the water for these ancient rivers, whose beds are now found far above the level of the present river system, and whose courses are generally crossed by the valleys of the modern streams. This condition of things continued long enough to permit the accumulation of beds of gravel, the gold-bearing alluvium, to a depth and extent unknown anywhere else in North America; and, if we speak of auriferous deposits, unequalled elsewhere in the world. Of the thickness of this accumulated material we have evidence in numerous places, where it has been protected from the action of subsequent denudation by a capping of volcanic materials. In many such places it reaches a thickness of 500 feet. Usually, however, it has been denuded to one half of this thickness, often less, and in many regions has been swept completely away.

Subsequent to the glacial and alluvial epoch to which the gold-bearing gravels are referred, there was a period of intense volcanic activity, the evidence of which is seen most conspicuously in the Table Mountains of California, so-called, which are cappings of basalt forming highly characteristic ranges, which portions of the ancient gold-bearing gravels are extensively explored in Tuolunine County by tunnels driven beneath the basalt cappings into the ancient river beds.

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* As given in Mr. R. W. Raymond's Official Report on Mines and Mining.

Following the outpourings of the volcanic matter there has evidently been an epoch of very active denudation by running water, which has broken up and removed the volcanic cappings, leaving them entire only here and there, as landmarks showing the ancient levels, and sweeping away also vast areas of the old alluvium and redistributing it as secondary or shallow placers at lower levels. This denudation was probably consequent on the sudden disappearance of the vast system of glaciers which up to that time crowned the entire range of the Sierras with ice. It was greatly more energetic in the southern portion of the Sierras than in the northern, where the mass of ancient alluvium remaining is much greater than it is in the former region. The extent of the ancient alluvium, as well as the energy of the power which produced it originally, and subsequently denuded it, becomes apparent on a study of the phenomena, carrying to the mind an overwhelming conviction.

It was these extensive deposits of gold which attracted the attention of the first adventurers, and were by them called "hill diggings;" but their real nature and significance were not at first fully understood, and, being generally much above any sources of water supply then available for washing, they received but little attention. Especially were they overlooked while the rich spoils derived from their secondary removal by denudation were available with no other means than the miner's pick, pan and shovel, upon those productive "bars" of adjacent rivers and in rich " gulches," where the gold (derived in large masses from the denudation of the ancient alluvium) lay open to the first comer in a concentrated form. So complete was the removal of the gravel in some of the southern counties that the gold, left behind by its weight, lay upon the naked rock, covered only by a few inches of vegetable mould—as at Mokolumne Hill, where, in the limits of a single claim 15 feet square, the precious metal to the amount of £10,000 fell, in one instance, to the lot of a single adventurer.

Gold in Australia.—From the researches of Mr. Selwyn, the Government geologist, it would appear that the gold-bearing strata of Victoria belong to a geological period much more ancient than that of the auriferous rocks of California. They belong, in fact, to the lower Palæozoic or Silurian epoch, and are (as Sir R. Murchison observed) almost identical in character with the Ural mountains. These rocks of Victoria are remarkable for the almost total absence of limestone bands, the number and extent of the quartz veins, and the constantly recurring protrusions, at short intervals, of granitic and occasionally of plutonic trappean rocks.

The quartz veins vary in thickness, from that of a sheet of paper to many feet; the thick veins being found in the lower portions of the series, but they are as a rule less productive than the thinner ones in the upper beds. The mines are frequently worked to immense depths.

Although the metamorphic rocks of Victoria appear to belong to a more ancient geological epoch than those of California, it should be observed that the drifts or deep diggings of the two countries seem to belong to the same geological epoch, and are both covered by a volcanic mass. Mr. Selwyn has concluded that there must exist two distinct series of quartz veins; the older ones having been formed previous to the Miocene period, and being comparatively barren; while the newer ones formed subsequently, but previous to the Pliocene period, are productive.

South Australia, Western Australia, Queensland, Tasmania, and New Zealand—especially the latter—are also gold-producing countries.

Chemical Geology of Gold.—Professor Phillips, in the Proceedings of the Royal Society for 1868, has given the following as the result of his examination of the geology of California, regarded from a chemical point of view:—

"(*a*) Quartz veins have generally been produced by the slow deposition from aqueous solutions of silica on the surfaces of the inclosing fissures.

"(*b*) From the general parallelism with its walls of the planes of any fragments of the inclosing rock which may have become imbedded in a vein, it is to be inferred that they were mechanically removed by the growth of the several layers to which they adhered, and that a subsequent deposition of quartz took place between them and the rock from which they had become detached. In this way were introduced the masses of rock known as 'horses.'

"(*c*) The formation of quartz veins is due to hydro-thermal agencies, of which evidences are still to be found in the hot springs and recent metalliferous veins met with in various parts of the Pacific coast.

"(*d*) From the variable temperatures at which the vacuities in their fluid cavities become filled, it may be inferred that they are the result of an intermittent action, and that the fissures were sometimes traversed by currents of hot water, whilst at others they gave off aqueous vapour or gaseous exhalations. This is precisely what is now taking place at Steamboat Springs in Nevada, where the formation of a vein is in progress, and from which currents of boiling water are often poured forth, whilst at other times the fissures give off currents of steam and heated gases only.

"(*e*) That gold may be deposited from the same solutions which give rise to the formation of the inclosing quartz, appears evident from the presence of that metal in pyrites inclosed in siliceous incrustations, as well as from the fact of large quantities of gold having been formed in the interior of the stems of trees, which in deep diggings are often converted into pyrites.

"(*f*) The constant presence of iron pyrites in auriferous veins, and when so occurring its invariably containing a certain amount of gold, suggest the possibility of this sulphide being in some way necessarily connected with the solvent by which the precious metal was held in solution. It has been shown that finely-divided gold is soluble in sesquichloride of iron, and more sparingly in the sesquisulphate of that metal. It is also well known that iron pyrites sometimes result from the action of reducing agents on the sulphate of that metal. If, therefore, sulphate of iron, in a solution containing gold, should become transformed by the action of a reducing agent into pyrites, the gold, at the same time being reduced to the metallic state, would probably be found enclosed in the resulting crystals of that mineral.

"(*g*) The silica and other substances forming the cementing material of the ancient auriferous river-beds, have probably been slowly deposited at a low temperature.

"It should be mentioned that Professor H. Wurtz differs from Sir R. Murchison as to the origin of gold, and has suggested a very plausible theory, ascribing it to a totally different cause. Having observed that aqueous solutions of sulphate of sesquioxide of iron were capable of retaining in solution a small but appreciable amount of metallic gold, and coupling this with the general belief that gold and gold-bearing sulphurets have been of aqueous deposition, he suggests that all the gold was held in solution in a 'Prozoic thoroughly oxidated ocean,' the sulphurets of iron now so abundant in crystalline rocks being dissolved in it as persulphates, and this persulphate in its turn being quite capable of retaining in solution all the gold in the earth's crust. When, however, in this lifeless ocean organic existence commenced, together with the accompanying deoxidising processes, the ferric became ferrous salts, and these being incapable of retaining gold in solution, it was deposited. Wurtz refers to one of the two richest gold-fields yet discovered—that of California—as disproving Murchison's general conclusion, that all gold-fields are of the Lower Silurian Age, the Californian gold-bearing formations being as recent as the Jurassic."

As a conclusion of the whole matter, one may say that the origin of gold is so far completely enveloped in mystery.

Scientific research has not as yet thrown sufficient light on the question *how* the gold originated in the rocks, and the chemical hypothesis above enunciated is an *hypothesis* only, without having (to many minds) any substantial evidence upon which it may be accepted as solid, demonstrable truth.

I have ventured, however, in the foregoing pages of this chapter, upon somewhat full illustrations of the intimate connection which undoubtedly exists between the ancient volcanic eruptions, and the accompanying thermal springs and solfataras, with the mineral lodes which are of so much interest at the present day. It is a matter of every-day proof that through those agencies the precious metal lodes have in some way been originated; and many contemporaneous formations of crystalline silicates, formed by the agency of thermal waters, are to be met with in various parts of the globe, which are the last remnants of a once grand and powerful volcanic activity.

CHAPTER XII.

RETROSPECT AND PROSPECT.

CAUSES OF FAILURE in Gold Mining.—MR. WILLIAM TOPLEY on the Future Production of Gold.

Causes of Failure in Gold Mining.—Although the causes of failure in gold mining are not properly within the scope of this work, I will venture to offer a few remarks upon that subject, seeing that at the present time it is a matter which widely concerns the public interest.

(1.) It is notoriously true of mining property generally, and not only of gold mines, that instead of dealing with such undertakings as we do with other business enterprises, they are too often made the object of stock speculations, in which "corners" are made by the unscrupulous, and the venturesome and unwary are again and again caught, as they ever will be, by their more vigilant and astute adversaries. These "stock speculations" are in plain language gambling, and a continuation of such gambling—with its resulting losses to many and gains to a few—leads to periods of depression in the mining market, during which time many promising mines have to suspend operations for lack of capital to develop them. Gold mines are then generally at a great discount. But all this has, it is needless to say, no necessary connection with legitimate mining, which should in no sense suffer disparagement from practices so detrimental to its best interests and so foreign to its objects. Still, many failures of legitimate undertakings are directly due to over-speculation.

(2.) The cases where large sums of money have been spent on extensive and expensive machinery before the investors have definitely ascertained that they have a veritable gold mine to deal with, can be counted by scores or hundreds, perhaps even by thousands. It is a safe rule to develop your mine first, before a penny is spent on mills, batteries, and other appliances for treatment of the ores.

(3.) Adopting wrong processes for the treatment of the produce is another fruitful cause of loss. It should never be forgotten that the fact of a certain process working well for the ores of one mine is no guarantee that it will do the same for others. Remember that a particular process is not necessarily suitable for every ore, though it may happen in individual cases that the ore is found adapted to a process selected more or less fortuitously.

(4.) Too small a percentage of the precious metal saved after heavy expenditure for extracting and raising ores is another cause of failure. This result may fall within the category indicated in the last paragraph; or it may be owing to excessive outlay in works and appliances, which have been estimated on too large a scale; or it may be owing to incompetent or defective management.

(5.) Again, investors may be misled as to the value and prospects of a gold mining property by high assays of samples. *The assay of picked samples never tells the value of a gold mine.* To determine the *average value* of the ores throughout a vein is a difficult and complicated matter, which is not to be determined by single assays, and many failures in promising speculations may be traced to high preliminary assays and low returns after the mills have been erected.

These cautions may seem to many to afford them very little help or guidance in particular cases in which they may desire to determine for themselves what is, or is not, a prudent investment. But no rules which would be sufficient for that purpose can be laid down. Those familiar with mining processes can only point out the special risks attaching to enterprises, the conduct of which is beyond the knowledge and control of the great body of investors who find the means of working them, leaving it to the investors themselves to shape their course in venturing upon undertakings which they cannot personally control.

The Future Production of Gold.—In a paper read before the British Association at Manchester, in 1887, Mr. William Topley, F.G.S., Assoc. Inst. C.E. (of the Geological Survey of England and Wales), dealt very carefully and exhaustively with the important question of the future production of gold, treating it with regard both to the geological distribution of auriferous ores and minerals, and to the actual yield of gold from the various sources of supply in former years. The whole paper (which deals in like manner with the future production of silver), may be commended to the consideration of all who are interested in the production of the precious metals; but I am glad to avail myself of Mr. Topley's permission to quote here the subjoined passages,* in which he summed up the conclusions at which he had arrived, so far as regards the production of gold:—

"In taking a general review of the goldfields likely in the near future to yield the most constant supply, it is evident that an important place must be given to Russia. With a very slight fall in the produce of Australia and of the United States, Russia would again take her old place at the head of the gold-producing countries. With its enormous areas of placer gold only partially worked, and its Siberian veins untouched, a steady yield of gold may be anticipated for many years to come.

---

* From the Report (1887) of the British Association for the Advancement of Science. Mr. Topley's paper was prepared at the request of the Section of Economics, and was ordered to be printed *in extenso* by the General Committee.

"The United States and Australasia have of late years been running very closely together, Australia being slightly in excess. In the former there is now a slight tendency to rise in yield.* A permanent rise cannot safely be anticipated; a more steady yield than in past years is all that can be hoped for: and this it seems likely may be the case, largely due to quartz mining. The rapid fall in the gold produce of the United States from 1877 to 1883 was chiefly due to the decrease of silver mining in the Comstock district, about forty per cent. of the value here being gold. If we deduct the silver gold, we see that the fall from 1877 was a very gradual one. The vast placer deposits of California, now in great part sealed by repressive legislation, will be to some extent again worked either by drift mining or by hydraulicking, with provision for the retention of the débris. Australasia shows a gradual steadying of the produce, neither placer nor quartz mining varying much from 1880 to 1885.

"Of the newer goldfields the first place should probably be given to Venezuela, &c. The wealth of this country in gold quartz is well established; but we may perhaps expect for a time a greater development of alluvial mining.

"South Africa is generally looked upon with favour as a source from whence our future supply of gold may in part be drawn. Without doubt there are here rich lodes, and it would be strange if this country were destitute of rich placers; though of this there is as yet but little evidence. From these sources mines may possibly be worked at a profit which will give a steady yield of gold; but there is as yet no evidence that the yield will be sufficient in amount to materially influence the world's production.

"As regards India the prospect is still less hopeful. That large quantities of gold were raised here by the native princes in times preceding the British rule is tolerably certain; but it is probable that this large production was spread over long periods of time, and certainly it was raised under conditions—of forced labour, &c.—which are not now applicable.

---

* Since this was written, the United States' statistics for 1886 have appeared. The yield of gold for the last four years is stated as follows: 1883, 30,000,000; 1884, 30,800,000; 1885, $31,801,000; 1886, $35,000,000. —W. T.

"It is unlikely that India will ever contribute to the world's stock sufficient native gold to materially influence the total production. A far more important point is the amount of gold hoarded in India, and the probability or otherwise of that being some day set free. Most estimates concerning gold are ludicrously vague, but on the question of the amount hoarded vagueness is unavoidable. It is known that £130,000,000 of gold has been taken into India since 1835; practically none of this is in circulation (silver being the standard and the coinage of India). How much was hoarded in the centuries preceding 1835 no one can say. If it only equals the amount hoarded since, we have £260,000,000, or nearly thirteen times the world's present annual production. The original source of this gold and the ways by which it reached India would be an interesting subject for inquiry. Since 1851 it is the gold of the world, mainly sent through England; but in the long past times it was probably in part of native production, in part the gold of Europe, sent over the old trade routes in return for the manufactured articles of India. It is supposed that at least as much silver is hoarded in India as gold. If so the value of silver and gold hoarded in India since 1835 nearly equals in value one-third of the total amount of gold and silver coin now in circulation in the world.

"Famines set free some of this gold, and we may perhaps anticipate that the diffusion of Western ideas will free more, but it is unlikely that gold will come from this source in sufficient quantities to influence the annual production of the world.

"British Columbia may possibly increase its yield. Other countries (as yet little known), as Equatorial Africa, Borneo, and North China, may add somewhat to the world's stock. A steady though comparatively small supply may be looked for from the treatment of silver ores and from the auriferous ores of other metals. In all parts of the world an increased supply is assured by improved methods of mining, milling, and metallurgy; this will be obtained by an actual increase from ore now worked, and also from ores of lower grade, made profitable by the improved methods; whilst tailings, &c., of former times can in some cases be profitably worked over again.

"But for all practical purposes the chief sources will probably continue to be the goldfields of the United States, of Australasia, and of Russia, aided by the development of the goldfields of South America. Everything points to a steady production from the three areas first-named, and to an increased yield from the last.

"In all metal-mining there is far too much gambling and wild speculation, and it is doubtful if, taking metal-mining all round, the value raised equals the expenses. But gold-mining cannot be ranked with ordinary mining. There is a glamour about gold which blinds men to ordinary prudential considerations. The wildest schemes meet with willing supporters, and money is always forthcoming to develop the poorest mines and to keep them going upon the most shadowy of hopes.

"Enormous fortunes have been made in gold-mining by a few lucky speculators. But for one who has been thus fortunate there are scores who have lost heavily.

"The facts which we have been considering as to the probable future of gold-mining warrant us in believing that the industry will gradually make for itself a sounder and more honest position. But there must ever be great uncertainty, and therefore a wide field for speculation and for dishonest dealing.

"If a steady and undiminished production of gold is essential for the well-being of the world, perhaps what we have most to dread is a sudden influx of common-sense and prudence in the investing public; for this would at once close a great number of mines, and might considerably diminish the world's production. But probably this contingency is sufficiently remote to be safely left out of consideration."

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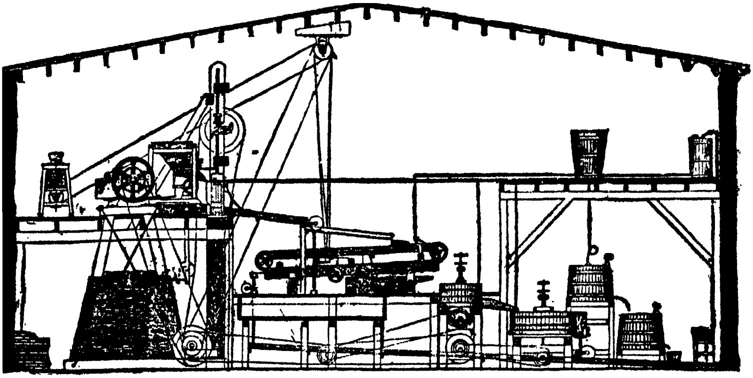

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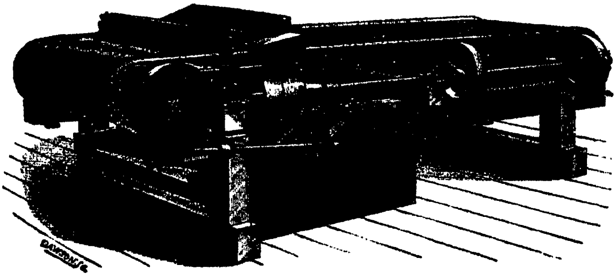

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